COAL DIVISION

1948

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(INCORPORATED)

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FOREWORD

This, the eleventh volume of Coal Division TRANSACTIONS, contains papers presented at Coal Division meetings subsequent to the 1946 annual meeting in Chicago. At the 1947 and 1948 annual meetings in New York, and at the 1946 and 1947 fall meetings in Philadelphia and Cincinnati, the attentive reception and discussion devoted to the texts of the papers herein presented attested to their pertinency and technical appeal. The Division's sincere and grateful appreciation is extended to George Keller, Chairman of the Program Committee during that period, for his unremitting activity in securing successful programs and to the Publications Committee, under Chairmen J. A. Younkins, Jr., and H. P. Greenwald, which assisted so materially in maintaining high technical and editorial standards.

Under the wise and skillful leadership of Chairmen Henry Hebley, in 1946, and Evan Evans, in 1947, the Coal Division continued to increase the services and usefulness to the coal industry evolved and expanded by preceding chairmen. All phases of this basic industry, from mining through the many varieties of utilization, are periodically brought to the attention of Division membership at technical meetings and by publication of permanent records of latest development, thus providing closer cooperation and understanding among the several constituent branches.

During the past year the Division has pointed out upon public record the paucity of realistic information concerning minable coal reserves within this country. A representative and active committee, with Henry Hebley as chairman, is cooperating closely with the National Bituminous Advisory Council in formulating a practical procedure for intensive and rapid reappraisal of reserves on which future mining needs may be intelligently appraised.

In answer to a general feeling that the functional organization of the Coal Division needed critical review, a special committee, with C. C. Wright as chairman, was appointed to survey this problem and to recommend specific methods for increasing the usefulness and scope of Divisional activities. The task assigned to this committee was promptly and admirably discharged, with its report being enthusiastically approved for adoption by the Executive Committee. This report and the enabling by-law changes will be presented to the Coal Division at the annual meeting in 1949 for adoption.

As the American Institute of Mining and Metallurgical Engineers embarks upon a reorganization and new publications program in 1949, the Coal Division is happy that one of its most distinguished members, Dr. L. E. Young, will serve as President. It is confidently expected that both Institute and Coal Division appeal to the mining profession will be enhanced and will continue to expand under the modified policies.

This foreword would be neither proper nor complete without recognition and acknowledgment of the consistently able work of our secretary, D. R. Mitchell, in guiding and performing the year-around activities of the Division. The gratitude and appreciation of the Division are likewise extended to the committee chairmen and members, the authors of technical papers, and the many active members who have contributed so greatly in promoting the affairs of the Division.

CLAYTON G. BALL,
Chairman, Coal Division

CHICAGO, ILLINOIS,
November 1, 1948

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Pittsburgh Coal Seam in Northern West Virginia

BY S. D. BRADY, JR.,* MEMBER AIME, AND W. D. STEELE†

(New York Meeting, February 1948)

THE Pittsburgh coal seam in West Virginia contains the largest coal reserves of any coal seam in that State and is, therefore, one of the most important seams, and attains minable thickness and purity throughout that part of the seam extending from the Northern Panhandle to Monongalia Co. thence south and southwest across the state to Wayne Co. near Huntington. The Pittsburgh coal is also minable in the Upper Potomac region but is now practically exhausted there. The total minable areas of this coal amount to 2,165 sq miles and are estimated, by the West Virginia Geological Survey, to contain 13¼ billion net tons. The largest body of Pittsburgh coal outcrops on the western edge of West Virginia, along the Ohio River as far south as Moundsville, West Va., going under the river at that point and extending at various depths down the river as far as Wetzel Co. line where it thins out to such an extent that it has little practical value, and on the eastern edge this large body outcrops along the Monongahela River.

The territory to be covered by this paper is the area known as the Northern West Virginia District comprising Wetzel, Monongalia, Marion, Preston, Harrison, Doddridge, Taylor, Barbour, Upshur, Lewis, Gilmer, Braxton and Calhoun Counties, and is also known as Dist. 3 under the Bituminous Coal Act. This region was originally opened for commercial mining along and adjacent to the Monongahela River around 1852 when the Baltimore and Ohio Railroad was constructed to Fairmont. It has been extensively developed, first, in 1880 (when the Monongahela River Railroad, now B. & O. R. R., was constructed) between Fairmont and Clarksburg, and later, in 1916, between Morgantown and Fairmont when the Monongahela Railway was constructed and placed in operation, this area furnishing the major portion of the production over the past years. With the mining out of a great deal of this territory adjacent to the Monongahela River considerable interest has been developed in the back acreage; and within the past few years a number of diamond core drill holes have been made in the back areas of Monongalia and Marion Counties. In this back area the oil and gas well drilling showed that the Pittsburgh coal seam was of minable thickness and ranged in depth from 350 to 1,000 ft below the surface of the stream valleys but this drilling did not furnish any definite data as to its thickness and quality.

ECONOMIC IMPORTANCE

The Pittsburgh seam in Northern West Virginia is of primary importance as a coal resource not only because of its large reserves but because of its contributing a very large part of the coal used in this country for general steam uses. The Pittsburgh seam is a consistently thick seam, averaging 8 ft in thickness, with very easy grades underground, so it has always been

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* Consulting Engineer, Summersville, West Virginia.

† Preparation Manager, Consolidation Coal Co. (of West Virginia).

TABLE 1—*Production and Value of All Coal by Counties*

N. W. Va. Counties	1933[a]			1943[b]			1944[c]			1945[d]			1946[e]		
	Net Tons	Value		Net Tons	Value		Net Tons	Value		Net Tons	Value		Net Tons	Value	
		Dollars per Ton	Tons per Man Day		Dollars per Ton	Tons per Man Day		Dollars per Ton	Tons per Man Day		Dollars per Ton	Tons per Man Day		Dollars per Ton	Tons per Man Day
Barbour	993,388	0.93	6.06	2,130,166	2.26	5.82	2,667,455	2.57	6.63	2,889,650	2.76	7.33	2,849,527	3.05	8.51
Braxton	26,543	1.02	5.47	29,531	2.39	2.77	20,421	2.61	3.05	23,998	2.85	2.95	73,996	3.03	8.34
Doddridge															
Gilmer	14,986	1.27	2.78	41,343	2.29	2.68	14,801	2.59	5.23	14,282	2.13	4.99	69,935	2.19	7.83
Harrison	3,146,594	0.92	6.56	9,082,887	2.42	8.45	13,007,442	2.59	9.83	12,831,076	2.80	10.09	11,019,788	3.05	11.98
Lewis	Included in Braxton			18,980	3.00	2.78	15,420	3.01	3.70	15,476	2.50	4.02	23,051	3.17	9.81
Marion	7,010,434	0.99	6.59	11,555,914	2.45	6.18	10,869,731	2.68	6.32	10,171,644	2.85	6.75	7,909,521	3.21	7.20
Monongalia	4,799,114	0.86	5.54	11,298,711	2.37	7.83	11,714,368	2.54	8.06	9,727,299	2.74	7.82	9,084,781	3.09	8.66
Taylor	1,126,360	0.89	5.12	1,128,053	2.48	6.52	1,355,943	2.76	7.25	1,408,175	2.91	8.14	1,913,914	3.51	6.54
Upshur	126,958	0.83	6.63	436,620	2.32	5.12	555,771	2.59	5.86	522,322	2.73	5.49	444,954	2.92	7.08
Wetzel															
Total—(WVA)	17,244,377	0.92	6.21	35,722,205	2.45	6.57	40,221,352	2.67	7.17	37,603,922	2.88	7.29	33,389,467	3.19	8.20
Total—(U. S.)	333,631,000	1.34	4.78	590,177,069	2.69	5.38	619,576,240	2.92	5.67	577,617,327	3.06	5.78	533,922,068	3.44	6.30

[a] U. S. Bur. of Mines—Minerals Year Book 1943.
[b] U. S. Bur. of Mines—Minerals Year Book 1944.
[c] U. S. Bur. of Mines—M.M.S. No. 1359.
[d] U. S. Bur. of Mines—M.M.S. No. 1468.
[e] U. S. Bur. of Mines—M.M.S. No. 1558.

one of the lowest cost regions in the United States. This 7 to 9 ft thick seam of coal is overlaid with from 1 to 3 ft of slickensided shale called draw slate which has always caused more or less trouble due to the ease with which it air-slacks. To offset handling this draw slate, which is done in Ohio and the thin vein regions of Western Pennsylvania, 10 to 18 in. of coal is left up to protect the roof, making a good roof for mining unless the entries are used too long. Retaining this 10 to 18 in. of coal still leaves from 6½ to 7 ft of minable coal which gives ideal working conditions and low cost operation compared to other regions.

The West Virginia Geological Survey Reports state there are estimated to be 9½ billion tons of Pittsburgh coal in Northern West Virginia but this is based on mining coal down to a thickness of 2 ft and is for 100 pct recovery. Adjusting these figures to a minimum thickness of 4 ft the estimated 100 pct recovery in this area is 8.3 billion tons. It is well known that a good recovery of Pittsburgh coal in the past was 70 pct of the minable thickness but during the last few years this recovery has been as high as 90 pct in a few instances. Using the 70 pct recoverable coal as a basis and at the present average production of 32 million

TABLE 2-A—*Production Pittsburgh Seam Only. W. Va. State Dept. of Mines Annual Report*

N. W. Va. Counties	1941		1942		1943	
	Deep	Strip	Deep	Strip	Deep	Strip
Barbour	1,465,064		1,598,867		1,604,054	248,431
Braxton	14,118		23,690		15,641	
Doddridge						
Gilmer	50,505		75,731		39,525	
Harrison	5,199,991		5,896,678	660,910	5,910,904	3,017,544
Lewis			20,149			
Marion	10,349,635		11,829,738	136,689	11,391,666	66,150
Monongalia	7,166,642	2,705	8,660,164	114,454	7,187,332	356,214
Taylor	572,711		819,078	100,257	962,894	113,316
Upshur						
Wetzel						
Totals	24,818,666	2,705	28,924,095	1,012,310	27,112,016	3,801,655

TABLE 2-B—*Production Pittsburgh Seam Only. W. Va. State Dept. of Mines Annual Report*

N W. Va. Counties	1944				1945				1946			
	Deep	Strip	Tons per Man Day Deep	Tons per Man Day Strip	Deep	Strip	Tons per Man Day Deep	Tons per Man Day Strip	Deep	Strip	Tons per Man Day Deep	Tons per Man Day Strip
Barbour	1,427,108	724,120	5.57	13.63	1,464,564	986,200	5.56	11.40	1,584,690	536,660	6.87	21.71
Braxton	6,229		1.45		20,625		2.35		7,569	56,916	3.00	47.58
Doddridge												
Gilmer					4,486		6.91		20,770	9,627	9.58	11.12
Harrison	5,902,591	6,629,025	7.07	15.92	5,608,801	7,023,780	6.91	20.67	4,396,499	6,050,563	8.13	26.30
Lewis					593		[a]		981	21,510	[a]	22.94
Marion	10,491,783	272,609	6.45	3.96	9,479,310	134,461	6.99	18.50	7,752,069	17,782	8.27	23.87
Monongalia	7,289,616	638,684	7.60	20.11	5,788,265	705,572	7.54	18.59	5,371,869	620,967	8.86	19.54
Taylor	916,769	253,949	5.51	21.51	817,705	239,535	5.50	24.22	1,103,564	660,425	6.07	26.68
Upshur					13,902	15,942	5.51	15.47	10,581	6,676	6.14	23.16
Wetzel												
Totals	26,034,096	8,518,387			23,198,151	9,015,490			20,248,592	7,981,126		

[a] All production from small hand mines. Data for net tons per man day not available prior 1944.

tons per yr of Pittsburgh coal from this field, there is a remaining life of 160 yr in the Pittsburgh seam.

TABLE 3—*Distribution All Coal from District 3 (Northern West Virginia)* 1945

Use	Net Tons	Per Cent	Movement	Net Tons	Per Cent
Industrial	20,894,745	49.3	Railroad	32,720,097	77.2
Retail	3,015,887	7.1	River	458,993	1.1
Railroad Fuel	13,165,250	31.0	Ex River	152,407	0.3
By-product Coke Plant	5,317,749	12.6	Tidewater	4,032,486	9.5
			Lake	4,414,299	10.4
Total	42,393,631	100.00	Truck and others	615,349	1.4
			Total	42,393,631	100.00

U. S. Bureau of Mines—M.M.S. No. 1444.

TABLE 4—*Sizes and Destination of All Coal Shipped from District 3 (Northern West Virginia)* 1945

Size	Net Tons	Per Cent	Destination	Net Tons	Per Cent
Lump and Egg	2,718,394	6.4	New England	1,125,772	2.6
Nut and Pea	7,385,392	17.4	N. J.-Pa.-Md.-Del	9,072,095	21.2
Modified Mine Run	24,811,298	58.6	Va.-D. C		
2-in. Nut Slack	3,224,818	7.6	New York	3,835,897	9.0
¾-in. Slack	4,242,478	10.0	Ohio	3,239,990	7.5
			West Virginia	1,716,277	4.0
Total	42,382,380	100.00	Mid-west	352,928	0.7
			Canada	672,994	1.5
			Railroad Fuel	13,165,250	31.0
			Tidewater	4,032,486	9.5
			Lake	4,414,299	10.3
			Truck and others	1,232,627	2.7
			Total	42,507,687	100.00

U. S. Bureau of Mines—M. M. S. No. 1444.

In order to show the economic importance of the Pittsburgh coal produced from this region the statistics published annually by the U. S. Bureau of Mines and the West Virginia State Dept. of Mines have been used as shown in Tables 1, 2, 3 and 4. The U. S. Bureau of Mines data do not make a separation of the Pittsburgh coal seam for this district but the West Virginia State Dept. of Mines did show this separation and from this information it is found that the Pittsburgh coal seam represents 80 pct of the production in the Northern West Virginia area.

Table 1 shows that there has been a substantial increase in the price received per net ton mined for coal mined in Northern West Virginia, increasing from $0.92 per net ton in 1933 to $3.19 per net ton in 1946. It also shows that there has been a substantial increase in the tons produced per man day, based on total men employed, from 6.21 tons per man day in 1933 to 8.20 tons per man day in 1946, or approximately 33 pct increase. This lowering of the cost of production has been brought about by the installation of mechanical loading underground as well as the substantial increase of strip mining of coal in Northern West Virginia. Some deep mines, where fully mechanized loading is used, have attained a figure of 13.0 tons per man day for over a year period and the maximum has not yet been reached as management and men become more efficient. Table 2-A and Table 2-B show the effect of strip mining, on the production of Pittsburgh coal only, from its beginning in 1941 up to and including 1946. The strip mining of the Pittsburgh seam was practically 28 pct of the production in 1946 but will now probably decline because the majority of the stripping in the past few years was done on the outcrop of the coal with very little cover and future operations will depend on the using of larger machine units working to much heavier cover.

Table 3 shows the use and method of transportation to the market as taken from the U. S. Bureau of Mines Marketing Survey Report for the year 1945, which year is a representative year. It is noted that 80 pct of the coal from Northern West Virginia is used for industrial steam purposes and railroad fuel and only 12.5 pct for by-product coke, practically all of which is now produced by captive mines and shipped by rail to destination, while only 7 pct goes to retail yards for domestic use. It is also noted in Table 3 that only 1.4 pct of the coal produced in this district is shipped via the Monongahela River, which is in contrast to the larger movement of 35 pct of the Western Pennsylvania production shipped on the Monongahela River.

Table 4 shows the distribution of the sizes made with more than one-half of the coal being sold as mine run or modified mine run (modified mine run being the mine run with the lump or double screened sizes of coal removed). This size coal is used by the railroads for fuel and by a number of the large industrial consumers. Also this table shows that the majority of this coal is shipped to destinations in the Middle Atlantic States with New York, Ohio and New England in the order named.

TABLE 5—*Reserves—January* 1, 1947

N. W. Va. Counties		W. Va. G. S. 100 Pct	S. D. B. Based on W. Va. G. S.	Mined Out Including 1946[a]
Barbour	7.66⅔	297,236,016	231,462,158	40,392,914
Braxton	3.5	311,369,074	242,452,869	6,006,835
Calhoun	2.0	103,261,594		
Doddridge	6.0	611,540,582	477,947,289	
Gilmer	3–4.0–5	452,187,648	352,187,648	2,476,360
Harrison	6.0	1,667,351,348	1,550,819,636	241,295,543
Lewis	1 to 6.0	865,501,655	594,367,488	368,588
Marion	7.0	1,824,089,959	1,635,387,891	306,036,272
Monongalia	7.0	1,507,144,037	1,370,673,770	129,149,251
Preston	7.0	18,455,500		953,978
Taylor	7.0	110,063,908	110,063,908	38,564,563
Upshur	4.0	63,339,725	63,339,725	350,360
Wetzel	6.0	1,592,414,208	1,327,011,840	
Totals		9,423,955,254	8,335,714,222	765,594,666

[a] W. Va. State Dept. of Mines Report.

In the past 10 yr a definite drive, with the full support of the present mine owners in the Northern West Virginia District, has been made to increase their industrial steam market and to this end the majority of the large utility plants installed in the past few years have been built so Northern West Virginia coal can be used. This is a logical step in the lowering of steam costs for the general economic needs of the U. S. for, while Northern West Virginia coals are high volatile, medium sulphur and low fusion coals, furnace design has been improved to such an extent that these coals can be burned very economically because of their comparatively lower cost of production in the Northern West Virginia field.

RESERVES OF THE PITTSBURGH SEAM

As of Jan. 1, 1947, the reserves of the Pittsburgh seam of coal remaining in Northern West Virginia are shown in Table 5, the first column being based on the West Virginia Geological Survey Reports, the second column showing 100 pct recovery with a minimum thickness of 4 ft and the third column showing the tonnage mined to Jan. 1, 1947.

A substantial amount of the above tonnage is tributary to the Monongahela River loading but none of the above acreage will meet metallurgical coke requirements of a sulphur content of less than 1.3 pct, but would contain a good grade of industrial steam coal which would probably move to markets along the lower Monongahela and upper Ohio Rivers, replacing tonnage that is now considered of metallurgical quality. It is a known fact that the reserves of low sulphur coal in Western Pennsylvania, while being substantial, are now limited and a definite effort is being made to prolong the life of these areas. To this end it is to be expected that a larger tonnage of coal, good only for industrial steam purposes, will move via the river to steam plants and other industries in the Pittsburgh and upper Ohio valley district.

CHARACTERISTICS OF THE SEAM IN NORTHERN WEST VIRGINIA

In Northern West Virginia the characteristics of the Pittsburgh seam are well known over a considerable area of the territory as substantial areas have been mined out adjacent to the Monongahela and West Fork Rivers; and while there are a number of mines along these rivers that still have a 20-yr life, the future developments of the Pittsburgh seam in the next 10 yr will probably move into the western parts of Monongalia and Marion Counties. There the thickness and other seam characteristics are similar to the area that has been mined out, except that the mining of this coal will require shafts or slopes to reach the Pittsburgh coal lying from 350 to 800 ft below the surface where the mining plants will have to be located. As a basis for discussion in respect to seam characteristics and analyses, the Pittsburgh seam in Northern West Virginia was divided into 8 general groups as follows 1. Monongalia-Preston Counties. 2. Marion Co.—low sulphur area. 3. Marion Co.—high sulphur area. 4. Harrison Co.—east of West Fork River. 5. Harrison-Doddridge Counties—west of West Fork River. 6. Wetzel Co. 7. Taylor-Barbour-Upshur Counties. 8. Lewis-Gilmer-Braxton and Calhoun Counties.

Within each general group there are local areas where the characteristics are not in close agreement with the group averages and variation of the seam thickness and other variations may seem large to some. Mr. H. N. Eavenson[1] says: "But in view of the enormous area covered by this deposit and by the tremendous quantity of it laid down originally, it is remarkable that its quality is so uniform. Many coal seams can show much greater consistency in some qualities than the Pittsburgh seam, but considering quality, thickness, and mining conditions together, few mineral deposits of any kind anywhere, can rank with it."

In the area adjacent to the Monongahela and West Fork Rivers, the thickness and analytical characteristics of the Pittsburgh seam have been extensively studied by the Consolidation Coal Co. (WVA), using channel samples and core drill records as well as railroad car samples from 22 mines and the results of this work have been averaged and used in this study. When the Coal Producers Board for Dist. 3 (Northern West Virginia) was organized an extensive program was completed for analyzing the coal by railroad car samples from all going mines so there are now available several hundred analyses taken by this Board, and representative analyses from these records were used in this study. Within the past few years 44 diamond core drill holes have been drilled in the western part of Marion and Monongalia Counties and the records have been made available by the land owners for this study. The West Virginia Geological Survey Reports analytical data were used, where none of the above data were available in certain areas.

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[1] References are at the end of the paper.

RANGE OF THICKNESS

Fig 1 shows the variation of the thickness of the various benches and slate bands of the Pittsburgh coal in the 8 general groups. In the Marion Co. group the low sulphur area and the high sulphur area have been combined into one section as there is slight variation in thickness in this region. The standard practice in Northern West Virginia is to sample the coal in 3 benches, (1) the top 10 to 18 in. of the coal seam, (2) the mining section, excluding the 2 slate bands and, (3) the bottom 6 in. of coal. The slate bands were analyzed separately in a number of the core drill holes and the records were available for this study. The practice of excluding these slate bands is done on the theory that this slate will be removed by hand picking or mechanical cleaning of the coal at the tipple. It is noted that the Pittsburgh seam, in this area, contains generally 2 thin slate bands from 3½ to 8 in. apart, which are approximately 4 ft above the bottom of the seam, with a third slate band sometimes showing. In Monongalia County the third slate band shows in the upper portion of the seam while in Marion Co. and Harrison Co. it sometimes shows between the regular slate bands. In Lewis and Gilmer Counties, the 2 regular slate bands have disappeared and only one band, running from 1 to 3 in. in thickness and of a cannel coal nature, appears. In Wetzel Co. a sulphur streak of ⅛ to ½ in. appears in the middle of the bottom bench between the slate bands and the pavement. The coal sections shown in Fig 1 also illustrate that when the Pittsburgh coal seam reduces its thickness from 8 to 4 or 5 ft, all of the reduction in thickness appears to come in the top bench. This is especially true in the western part of Harrison Co., and Upshur, Lewis and Gilmer Counties.

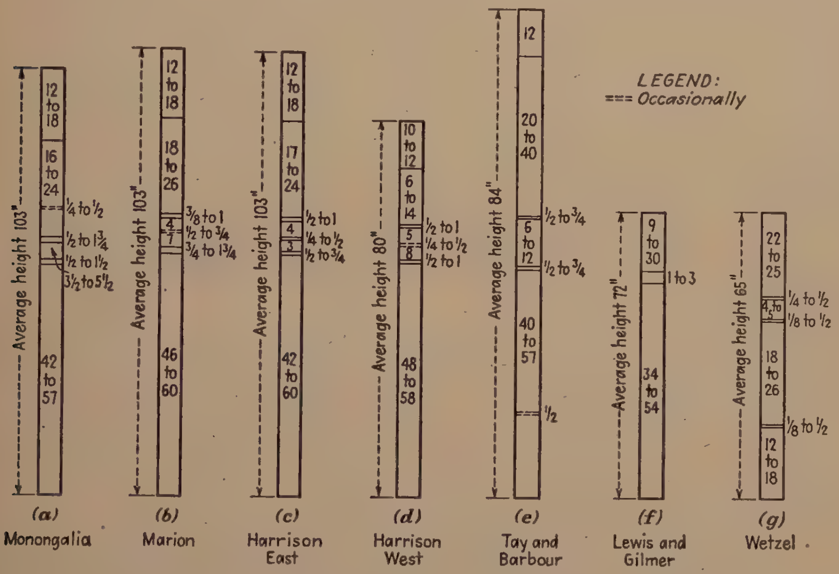

FIG 1—RANGE OF THICKNESS OF BENCHES AND SLATE IN PITTSBURGH COAL.

RANGE OF ANALYSES

Table 6 shows the range of analyses taken from large and small producing mines in all of the general groups except Wetzel Co. There are no suitable analytical data on Wetzel Co. coal which could be used in this study, so Wetzel Co. was omitted from the table.

TABLE 6—*Range Analyses of Pittsburgh Coal in Northern West Virginia District*

	M	VM	Ash	Sul.	Btu	Fusion	
Monongalia							
Av.	1.8	37.0	8.50	3.0	13,800	2,200	15 Mines
Max.	3.3	39.2	9.40	3.6	14,362	2,320	
Min.	0.6	35.8	8.05	2.1	13,370	2,110	
Marion H.S.							
Av.	1.40	37.0	7.00	2.2	13,800	2,175	14 Mines
Max.	1.66	39.0	8.52	2.4	14,360	2,290	
Min.	1.20	35.5	5.50	2.3	13,600	2,150	
Marion L.S.							
Av.	1.3	38.4	5.50	1.2	13,840	2,285	5 Mines
Max.	2.0	39.0	7.00	2.2	13,930	2,345	
Min.	1.1	37.5	6.2	0.76	13,784	2,090	
Harrison East							
Av.	2.1	38.9	6.0	1.7	13,946	2,125	5 Mines
Max.	3.1	39.6	7.1	2.9	14,012	2,175	
Min.	1.0	37.9	5.4	1.3	13,746	2,100	
Harrison West							
Av.	1.5	40.0	7.2	3.0	13,540	2,050	20 Mines
Max.	2.9	41.2	9.0	4.3	13,900	2,080	
Min.	1.0	38.8	7.0	2.8	13,400	1,990	
Barbour-Taylor-Upshur							
Av.	1.5	38.0	7.3	2.28	13,800	2,170	24 Mines
Max.	2.1	39.7	8.2	3.20	14,260	2,280	
Min.	1.1	36.3	6.5	1.70	13,310	2,060	
Lewis-Gilmer-Braxton							
Av.	1.5	40.0	6.5	2.40	13,540	2,150	34 Mines
Max.	1.8	43.67	7.35	2.71	14,071	2,200	
Min.	1.2	38.63	4.28	1.96	13,337	2,120	

Bituminous Coal Producers Board District 3—Analysis Railroad Car Samples, West Virginia Geological Survey County Reports.

TABLE 7—*Range of Analyses of Drill Holes in West End Monongalia and Marion Counties, West Virginia*

	M	VM	FC	Ash	Sul.	Btu	A.S.T.
Monongalia							
Roof Sect.							
Av.	1.00	34.50		15.2	3.50	12,050	2,250
Max.	1.30	40.40		16.92	4.74	13,200	2,540
Min.	0.70	32.90		9.6	3.23	11,500	2,050
Mining Sect.							
Av.	1.40	37.00		6.7	2.20	13,900	2,280
Max.	1.80	39.40		8.6	2.63	14,360	2,390
Min.	1.20	35.70		5.2	1.90	13,200	2,160
Marion							
Av.	1.10	39.30		16.5	3.90	13,100	2,160
Roof Sect.							
Max.	1.30	43.90		19.9	5.80	13,600	2,430
Min.	0.90	37.00		7.6	3.50	12,670	2,030
Mining Sect.							
Av.	1.30	39.80		5.7	2.50	14,200	2,140
Max.	1.50	43.60		7.8	3.90	14,360	2,290
Min.	1.10	37.50		5.5	2.30	13,800	2,065
Monongalia-Marion							
Slate bands							
Av.	0.61	28.00		32.5	1.50		2,715
Max.	0.81	38.30		75.2	2.30		2,810
Min.	0.23	23.50		22.4	1.20		2,395

Based on drill hole records from private sources.

Table 7 shows the range of analyses of the Pittsburgh coal cores removed from the drill holes in the western end of Monongalia and Marion Counties. This table shows the analyses divided into roof section and mining section. The roof section comprises the 10 to 18 in. of high ash coal normally left as roof coal in the coal mines of this area. These analyses prove that this roof coal is very high in ash and sulphur. The analyses of the mining sections shown in Table 7 were taken with the slate bands excluded and the analyses of the slate bands are shown separately. The analyses of the slate bands show that, if they had been included in the analysis of the mining sections, they would have had very little effect on the final results. The average thickness of coal included in the mining sections was 80 in. and the average thickness of the two slate bands would not exceed 2 in., therefore, the slate bands would only increase the ash by not more than ²⁄₁₀ above the pct of ash shown. If the slate bands had been included in the samples of the mining sections it would have a very slight effect of lowering the sulphur percentage and increasing the fusion.

A map showing the location of the mines and core drill holes where samples were taken, indicates that there is only a small area in Northern West Virginia that could be considered a low sulphur metallurgical coal, or coal having 1.3 pct sulphur or less. It also develops that there are several small "islands" of low sulphur coal scattered through the area but they are so small in area that they are not shown. The reserves of low sulphur coal remaining in Northern West Virginia, with one or two small exceptions, are practically now all owned by captive interests, therefore there is no substantial area of low sulphur coal remaining in this region that might be available for development.

From a study of the above-mentioned map in relation to the individual coal analyses, it develops that the volatile content of the Pittsburgh coal increases going west from the Monongahela River as shown by the analyses in the western end of Marion and Monongalia Counties, and the volatile increases further in Wetzel Co. as one approaches the Ohio River, assuming Wetzel Co. Pittsburgh coal will analyze similar to Marshall Co. coal. The volatile content also increases going south of Clarksburg.

THE EFFECT OF THE MAIN FAULT (WANT)

There appears to be an old stream channel located in the Pittsburgh coal seam running into Northern West Virginia from Western Pennsylvania that more or less parallels the Monongahela River until it reaches a point near the mouth of Indian Creek where it swings in a westerly direction along the Monongalia-Marion Co. line and extends for some distance in that direction as has been proven by the mining operations along it. This old stream channel is locally called a fault but it is actually a "Want," and appears to have been caused by an old stream eroding portions of the coal seam in the old days, geologically, and later filling the series of intrusions of varying sizes with sand which later under pressure became sandstone. The intrusions are of a varying size running from 50 to 200 ft through and being from 50 to 600 or 700 ft long and as the coal seam approaches the "fault" area the draw slate over the coal disappears and sandstone comes next to the coal making the roof.

There are indications that the Pittsburgh coal changes in some of its characteristics at this faulty area. The coal north of this line of "fault" is softer in structure. There are also from one to three bone, rock or sulphur bands appearing within the top 30 in. portion of the seam. The typical Pittsburgh bands normally near the center of the seam begin to lose their identity and often only one of the bands is discernible.

There is also a change in the coal analytically. From channel samplings north and south of the "fault" area the following comparisons were made:

NORTHERN MINE	5 IN. × 0	SOUTHERN MINE
0.64	Moisture	0.57
37.21	Volatile Matter	40.10
52.88	Fixed Carbon	51.36
9.27	Ash	7.97
3.12	Sulphur	3.15
13671	Btu	13950
2270°	A. S. T.	2200°
21.75	Fe_2O_3 in Ash	27.30

It is shown that on the coals of almost identical sulphur content that the volatile content is lower in the northern coal. The ash is higher and the Fe_2O_3 in the ash is lower in the northern coal. The ash softening temperature is slightly higher in the northern coal.

Additional comparisons were made after Float and Sink tests, on the 5 × 0 size after crushing the plus 5-in. coal.

FLOAT AND SINK TESTS
AT 1.45 SP. GR.

NORTHERN MINE	5 IN. × 0	SOUTHERN MINE
0.61	Moisture	0.47
38.26	Volatile Matter	41.31
53.26	Fixed Carbon	51.91
7.30	Ash	6.31
2.55	Sulphur	2.54
14119	Btu	14285
2520°	A. S. T.	2200°
16.98	Fe_2O_3 in Ash	22.75

FLOAT AND SINK TESTS
AT 1.60 SP. GR.

NORTHERN MINE	5 IN. × 0	SOUTHERN MINE
0.65	Moisture	0.46
37.50	Volatile Matter	40.87
53.72	Fixed Carbon	51.76
8.13	Ash	6.91
2.74	Sulphur	2.65
13932	Btu	14177
2510°	A. S. T.	2200°
17.06	Fe_2O_3 in Ash	23.27

INDICATED IMPROVEMENT
AT 1.45 SP. GR.

NORTHERN MINE		SOUTHERN MINE
21.25	Pct reduction in Ash.....	20.83
18.27	Pct reduction in Sulphur.	19.37
250°	Actual increase in A. S. T.	none

INDICATED IMPROVEMENT
AT 1.60 SP. GR.

NORTHERN MINE		SOUTHERN MINE
12.30	Pct reduction in Ash.....	13.30
12.18	Pct reduction in Sulphur.	15.87
240°	Actual increase in A. S. T.	none

A study of the detailed Float and Sink data shown indicates that the coal in the southern area can be cleaned to yield a comparative reduction in ash and sulphur content with that from the northern area. The iron oxide in the ash shows a proportionate reduction in each instance although there is no increase in the ash softening temperature of the coal from the southern mine there is an increase of 250° in the coal from the northern area.

EFFECT OF SELECTIVE MINING AND OTHER PRACTICES OR QUALITIES

Messrs. Davis and Griffin[2] say, in relation to selective mining:

"The problem will require careful study for each mine and will involve the study of mining methods, face-preparation practices, and selective mining in their relation to the improvements in analytical properties that may be effected by coal cleaning. Only by coordinating the underground practices and the coal-cleaning program will it be possible to obtain the maximum economical reduction in ash and sulphur.

"The following factor will each have its effect on the analysis of the coal produced:

1. Selective mining in respect to areas of various levels of quality.
2. Selective mining in respect to the proportion of the seam mined.
3. The method of loading employed.
4. The method of cutting and/or shearing.
5. The method of breaking down the coal.
6. The amount of refuse loaded with the coal."

Selective mining as practiced in some areas will improve the quality of the coal mined but in the Pittsburgh seam in Northern West Virginia, except in the low sulphur area, there is not much opportunity for any practical method of selective mining except to leave up the roof coal and, where possible, to leave down the bottom 6 in. of coal.

It has been found in Northern West Virginia, that, leaving the bottom coal in the mine during the old hand loading days was conducive to loading a high quality coal, but with the introduction of mechanical loading, particularly with the tractor mounted type, the soft bottom underlying the Pittsburgh seams permits this coal to be broken up and a large portion is now being loaded, therefore no separation was made for this coal in the analyses quoted in this paper.

Since the future of the Northern West Virginia coal field is wrapped up in the production of industrial steam coals at low cost, and since the quality of Northern West Virginia coal is such that, except for certain local areas, none of it is of metallurgical quality, it appears that it is impracticable to do anything more than is now being practiced, in either selective mining or bin mixing. Some efforts have been made, by the selection of sizes at the tipple and conveyor mixing, to try to reduce the ash content and thus increase the Btu.

MECHANICAL CLEANING

A number of mechanical cleaning plants have been built in this field, since the introduction of mechanical loading, for the purpose of securing a uniform ash in the coal shipped. In Table 8 is given a list of mechanical cleaning plants that have been built in Northern West Virginia in recent years and the analytical results being obtained show that some reduction in the ash content in the raw coal is being made along with producing a clean coal with uniform ash content.

TABLE 8—*Mechanical Cleaning Plants in Northern West Virginia*

Company	Mine	Type	Sizes, Inches
Canyon Coal and Coke Company	Canyon	Jeffrey Jig	4 to ½
Consolidation Coal Co. (W. Va.)	Arkwright	Chance Cone	5 to ⅜
Consolidation Coal Co. (W. Va.)	Owings No. 32	Chance Cone	5 to ⅜
Eastern Gas and Fuel Associates	Federal No. 3	Belknap Chloride	Egg, nut, pea
Jamison Coal and Coke Co	Jamison No. 9	Jeffrey Jig	5 to ⅝
Kellys Creek Collieries Co	Maiden	McNally-Pgh-Baum	5 to ½
Monongahela Rail and River Corp	Emily	Hydro-Separator	5 to ⅜
Pursglove Coal Mining Co	Pursglove No. 15	Chance Cone	5 to ⅜

With the use of loading machines there is no selective picking of the coal at the face and in order to secure higher tonnages per loading unit the coal must be shot up into smaller sizes, which greatly increases the problem of hand picking. Also with the mechanical loader there is great danger of having roof shales or bottom fire clay loaded with the raw coal where the supervision is not almost perfect. The great increase in mechanical loading in Northern West Virginia has been during the past ten years and all of the mechanical cleaning units have been installed during that time.

It will be noted in Table 8 that the washing plants do not wash any sizes below ½ in. If sizes below ½ in. were washed the cleaned coals would show an increase in moisture content that would offset the reduction in ash and result in little improvement in the ultimate Btu value. Since the greater proportion of the production from Northern West Virginia is for industrial steam use most purchasers of the coal contend that a 1 pct increase in moisture is equivalent to 1 pct increase in ash for their purposes. For this reason only the sizes above ½ in. or ⅜ in. are washed. If larger size slack coals are desired the washed sizes are reassembled with the dry slack and the average quality improved.

Drying plants for drying this coal can be used but again the question arises as to whether mechanical cleaning and drying, with its investment and operating cost, is practicable.

SUMMARY AND CONCLUSION

The Pittsburgh coal seam in Northern West Virginia is of great importance to the economy of the United States as it has, in the past few years, contributed approximately 6 pct of the total bituminous tonnage and, in the future, it is expected to increase percentagewise owing to its comparative low cost of production, being deep mined much more cheaply than the steam coals of Western Pennsylvania and Ohio.

The recoverable Pittsburgh seam of coal in Northern West Virginia as of Jan. 1, 1947, is estimated to yield 5 billion tons, or about 160 yr of life at the recent production rate.

The physical characteristics of the Pittsburgh seam of coal in Northern West Virginia indicate that about 80 pct of the reserves in Marion, Monongalia, Harrison, Barbour and Taylor Counties will show at least 6½ ft of minable height, while in Braxton and Gilmer Counties some local areas show a minable height of equivalent thickness but the majority of the reserve coal in the balance of the area will probably have a minable height of 5 to 6 ft.

The analytical characteristics of the Pittsburgh coal in Northern West Virginia indicate that 98 pct of the reserve tonnage will be a good grade of industrial steam coal and that the remaining 2 pct, or 100 million tons, will be of metallurgical grade containing 1.3 pct, or less, sulphur.

There is a definite change in the analytical characteristics of the Pittsburgh seam of coal as between the northern and southern side of the "fault" or Want, or between Monongalia and Marion Counties which is shown by the lower volatile and higher fusion and lower Fe_2O_3 content of the ash in Monongalia County.

Analyses of the Pittsburgh coal reserves of Dist. 3 or Northern West Virginia clearly indicate that selective mining, as practiced with the leaving of roof coal and possibly the bottom coal, will improve the ash and sulphur content in the final product.

Mechanical cleaning of screened sizes is an economical necessity where mechanical loading is practiced in the Pittsburgh coal in order to secure lowered cost of operation and a uniform lowered ash product.

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DISCUSSION

(H. F. Hebley and E. E. Finn presiding)

G. H. Ashley*—A few years ago we finished a survey of Fayette Co., Pa., where they commonly estimate that they are getting from 85 to 95 pct of the Pittsburgh coal bed, and in making our estimate of the reserves, 90 pct was used. But if the mine maps are consulted and one figures out the area that is actually mined out compared with the tonnage obtained, it appeared to be only 63 pct. I think that is because of the 18 in. min. thickness used. When we figure our reserves and consider all those things, we get quite a different figure from that obtained by taking 90 pct of the estimated coal in the ground.

H. F. Hebley—That is what we have been trying to lead up to, to be realistic on what actual coal we are obtaining. There is also implicit in this paper the earmarking of metallurgical coal for metallurgical uses and the use of steam coal for steam uses.

That is a matter of economics and sometimes in some countries it is a matter of government control; not that I am advocating that, but it shows the trends toward which the whole world is moving.

L. E. Young†—This paper is an important and timely contribution to the published data on the Pittsburgh Seam in Northern West Virginia and it presents in convenient form much valuable information. This field has been an important producer for many years, but with the depletion of large coal areas in Southwestern Pennsylvania, it will play an even more important part in the future.

As has been pointed out by the authors, the physical conditions are such that the field is well suited to mechanical loading and full advantage is being taken of these favorable conditions by enterprising operators. The area of low-sulphur metallurgical coal is not large, but with selective mining and careful preparation, a satisfactory grade of low-cost industrial coal is being produced.

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The proposed reappraisal of the coal reserves of the U. S., now under consideration, is most timely. This paper is an important contribution in starting this work.

S. D. Brady, Jr. (authors' reply)—The Tables of Analyses are averages of many analyses and it was quite a job to select the ones that were truly representative of each area.

In regard to Dr. Ashley's and Dr. Young's remarks I wish to say that most of the present estimates of the reserve of Pittsburgh coal are upset by the fact that, in mining the Pittsburgh coal, it is necessary to leave roof coal to protect the draw slate; and if the working places are left open without immediate pillaring operations, within a period of 1 to 1½ yr this roof coal and draw slate will fall and make it necessary to split the pillars in order to recover them. This method causes a loss of at least 20 pct more of the recoverable coal before pillar mining is completed.

How Much Coal Do We Really Have? The Need for an Up-to-date Survey

By Andrew B. Crichton,* Member AIME

(New York Meeting, February 1948)

The oft repeated statements of the United States Geological Survey and the Bureau of Mines that the coal reserves in the United States are sufficient for 3000 yr have given us all a sense of security hard to disturb. If we discount that statement 66 pct, which to the uninformed seems drastic, we would still have 1000 yr supply, and why should mortal man worry about the difference between 1000 and 3000 yr supply of coal. That is exactly the attitude which many who should be better informed have taken toward the problem. A recent brochure "How Strong is America," by the New York Herald Tribune, states there is "no end to coal * * The truth is staggering * * U. S. has enough coal to last 4000 years." What the statement actually means is that the calculator got as high as 4000 yr and decided it was pointless to go on. The President's ERP Committee on Natural Resources and others in high places continue to assure the country of our superabundance of coal.

In the past few days the "Wall Street Journal" carried an item from Director Boyd of the Bureau of Mines saying that within a hundred years we would need to triple our annual coal production of 600,000,000 tons, making it 1,800,000,000 tons annually, and at that rate we had coal reserves sufficient for 1000 yr. In my humble opinion, we cannot now or ever produce 1,800,000,000 tons annually, because we do not have the reserves in the producing districts to do it.

Past reckoning in geology may be, as in the Psalms, a thousand years are as yesterday, but the future must be considered more realistically, and "a thousand years" is really a "heck" of a long time.

In most developed areas, mining engineers can count the coal reserves in terms of a hundred years or less, and in some important areas now producing large annual tonnages in much under fifty years. They think when they hear of thousands of years coal reserves, reference is made to distant fields and "the fields are always green far away."

The estimates of our coal reserves upon which we are relying were made in 1907 under the direction of Marius R. Campbell, Director of the United States Geological Survey. It is apparent this work was not well done. It was necessarily based on rather meager data because of the little development and prospecting as of that date. There were mistakes in the correlation of seams, and they assumed the continuity of thick seams throughout large coal bearing areas where coal often did not exist. The overstatement of coal bearing areas in the Campbell figures are common. The 1907 survey includes all coals over 14 in. in thickness to a depth of 3000 ft.

The total coal reserves according to Campbell were 3157.2 billion tons. The production and mining losses that year would amount to 570 million tons and

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indicate reserves sufficient for 5500 yr. There have been a number of estimates of our coal resources, some of them based on the Campbell figures less subsequent production, with liberal allowance for mining losses, but apparently no new base survey. The Wrather statement as of 1936 shows a total of 3180.8 billion tons, a slight increase over the Campbell figures, but it included certain radical increases in some and reductions in other state figures.

In the 1936 figures, Colorado tonnage is reduced 55 billion, Illinois 42 billion, Michigan 10 billion, New Mexico 143 billion, Oklahoma 24 billion, Pennsylvania 16 billion, Utah 103 billion and West Virginia 83 billion tons; while many were increased substantial amounts. Indiana was increased 8 billion tons, Kansas 22 billion, Kentucky 17 billion, Missouri 43 billion, Montana 78 billion, North Dakota 100 billion, Washington 43 billion, and Wyoming 196 billion tons. It is hard to understand these increases in view of subsequent developments.

The United States Coal Commission made a survey as of 1923 of recoverable reserves, and The National Coal Assn. has brought that survey down to 1945 by deducting the production meantime. They have reduced all the state figures and dropped some entirely, making in 1945 the total coal reserves at 1614.8 billion tons, or sufficient say for 2000 yr.

It is recognized that at best, in the calculation of coal reserves, there are many imponderables, but adding 100 billion tons or deducting 100 billion tons from a state's reserves with such reckless abandon would seem to indicate a degree of guessing hardly to be expected of our Government officials.

Some of the State Geological surveys have made more recent estimates of coal reserves based on later and more accurate information greatly reducing their state's figures, but the United States Geological Survey and the Bureau of Mines ignore them and continue to quote their own fantastic and misleading figures.

In *Pennsylvania* the State Geological Survey estimated coal reserves as of 1926 at 44 billion tons of recoverable coal. The coal Commission figures reduce this to 32 billion tons Jan. 1, 1945. The United States Geological Survey shows 14,200 sq miles of workable coals, which is the total area of the townships or counties that contain any coal. The State shows only 8300 sq miles of coal bearing area. They both indicate coal over the entire area 8½ ft thick, which to anyone familiar with the field is a gross overstatement. The State's figures show 537 million tons of Moshannon coal in Clearfield Co., and except for the dregs, this seam has been worked out for 20 yr. The total recoverable coal in Clearfield Co. was given as over 2 billion tons, and the whole area is about exhausted. The State survey indicates over a billion tons of recoverable Pittsburgh seam coal in the famous Connellsville-Greensburg field, and except for a few large operations nearing exhaustion, those fields are about worked out. Except for Greene and Washington counties, there are few large areas of undeveloped coal. In my opinion, 10 billion tons would be a high estimate of all the coal in all seams that can be economically recovered in Pennsylvania.

In *Maryland* the United States Geological Survey shows 8 billion tons in 455 sq miles, equivalent to a seam 18 ft thick over the entire area, while a great deal of this area contains no coal. I am well acquainted with the coal fields and coal mines of Maryland, and in my opinion 200 million tons is a high estimate of coal reserves.

In *Ohio* they show 86 billion tons, raised to 92 billion in 1936, while the State Geological Survey shows about 10 billion tons of recoverable coal. Mining losses in Ohio are high, due to roof conditions in the principal seams. Probably not more than 60 pct of the coal is being recovered. The best and cheapest mined coals are gone. From information available, 6 billion tons for Ohio is believed high, as over half of the reserves are in seams of extremely doubtful quality and thickness.

The *Illinois* figure of 240 billion tons would indicate a seam 7 ft thick over about two-thirds of the area of the state. The Coal Commission reduces this figure to 53 billion tons. Probably not more than 50 pct of the valuable 10-ft thick seam in southern Illinois is being recovered, and 12 yr ago it was said to have a distinctly limited life. Reports presented at a recent Illinois Coal Research Conference would indicate the original estimates of the United States Geological Survey were so overstated as to be useless. I have discussed Illinois coal resources with operators and engineers with varying results. Some southern Illinois operators are expecting substantial reserves in other fields they know little about. They may be disappointed. My guess for the recoverable coal in Illinois would be less than 20 billion tons, or 10 pct of original estimates.

What is said about Illinois could be applied with equal force to *Iowa*, *Missouri* and *Kansas*. A leading coal operator of that area, who is interested in the subject, says, instead of 29 billion tons, Iowa should be credited with 3.5 billion, Kansas 0.8 billion instead of 29 billion, and Missouri 3.5 billion instead of 84 billion tons.

Who would ever suspect that 11,000 sq miles, or one fifth of the state of *Michigan* contained coal in the amount of 12 billion tons? This the 1936 report reduced to 1.9 billion tons and by the Coal Commission to 100 million tons, but from what the leading operator says, and he has done considerable diamond drill prospecting, coal of commercial value is very limited.

Kentucky is credited with from 60 to 121 billion tons covering 16,670 sq miles in the east and west Kentucky fields. The high figure is equivalent to a seam about 7 ft thick over two-fifths of the state's area. The leading operator says about five billion tons or 50 yr work at the present rate of production is about all that is in sight.

West Virginia, the country's largest producer of coal, is credited with 17,000 sq miles (or ¾ of the state's area), and 231 billion tons, the equivalent of a seam 14 ft thick over the entire area. The Coal Commission shows 64 billion, but the State Geological Survey claims 116.6 billion tons, and they have probably done more work on coal resources than any other state agency. The state's figures would represent a seam 7 ft thick, and any one acquainted with the West Virginia fields would recognize this as an extravagant claim because of the very large areas within the coal fields where no coal exists. According to reliable engineer estimates, the famous New River and Winding Gulf fields are well on toward exhaustion. Some of the larger producers have but a few years' life and few more than 20 yr, and yet the state survey credits these areas with over 6 billion tons, or over 200 yr life. Few in the Pocahontas fields have reserves for more than 75 yr and most will be exhausted long before 50 yr, and yet the state's figures indicate 400 yr life. In nine coal producing counties of southern West Virginia, from which over 60 pct of the production now comes, the state shows 48 billion tons, while reliable private surveys show a little over 9 billion tons. In two northeast counties the state shows over 1.7 billion tons, while private estimates indicate but a trifling amount. In a newly developed field in the central area, comprising three counties, the state figures show over 13.5 billion tons, while private figures based on a great deal of diamond drill and crop prospecting shows a little more than a billion tons. It would appear from information available that West Virginia coal tonnage is not 10 pct of the Government's 1907 figures, or about 20 billion tons.

Virginia is shown to contain 1550 sq

miles in the southwest fields, with 21 billion tons, or the equivalent of a seam 14 ft thick over that area, which seems a very extravagant claim. The Brushy Mountain anthracite field, credited with 200 sq miles, or 128,000 acres, equal to a 4.7 ft seam over that area, will not surprise you more than the statement that the Richmond field has 96,000 acres and 600 million tons, the equivalent of a 4 ft seam over that entire area.

The *Alabama* field of 843 sq miles and 68.9 billion tons is the equivalent of an 8.5 ft seam over the entire area. It is said the Coosa field shown to contain 260 sq miles and 2.4 billion tons has little if any coal of commercial value. A prominent Alabama geologist well acquainted with the coals of that state says 3 to 6 billion tons would be a high estimate for that state.

One may well wonder where a billion tons of coal would be found in *California* and 900 million tons in *Georgia.*

Tennessee is credited with 4400 sq miles of coal over 8 ft thick, containing 25.7 billion tons, while the thickest seam worked is 5 ft and most coal is around 3 ft thick.

Oklahoma is shown to have 10,000 sq miles of coal 8 ft thick, containing 79.3 billion tons, which will surprise Oklahoma operators, as they mine coal only 17 in. thick and the average seam thickness in the mines of the state is 33 in.

Texas is said to have 8200 sq miles or 8 billion tons of bituminous coal and 200 sq miles of lignite, containing 23 billion tons, or equivalent to a 12 ft seam. One of the largest Texas oil companies drilled extensively in an area supposed to contain coal, but found none. The job seems to be to find these vast reserves, existent apparently only in the official reports. The impression in the east is that these big coal reserves are to be found in the west, and the Westerners assume they are in the east.

Colorado is shown to have 10,130 sq miles of coal containing 371.8 billion tons, indicating coal of 39 ft thickness throughout this entire area. If the 4180 sq miles of *possible* coal areas is included, the coal would be 27 ft thick. In the Uinta field of 6000 sq miles and 271.8 billion tons, the coal would be 47 ft thick. Does anyone imagine nature has blessed Colorado so extravagantly? Well, ask the coal operators of Colorado. The Chief Engineer of the largest operator in Colorado, a company owning tremendous coal acreage on the west slope and the east slope, and which company has pretty well covered the state looking for coal of special quality, says that 1 pct of 319 billion tons, or only 3 billion tons, for Colorado would be hard to find. "The coal deposits in Colorado are rather freakish and irregular. A six foot seam may disappear entirely a mile from the opening. The better grade coals of the Trinidad fields, which have been operating for sixty years, are about gone."

In *Utah* the 1907 report showed 196 billion tons. The 1936 report reduces that figure 103 billion tons to 93 billions. Where it all disappeared is not clear. The original figure indicated coal over 15 ft thick over the entire 13,000 sq miles. The Coal Commission gives 46 billion tons, and the leading coal operator says there are under 2 billion tons that can be recovered economically. These look like a series of guesses, and from my own experience I would consider the 2 billion figure the most reliable. Western mining engineers say the better grade coals in southwest Wyoming and Utah are largely worked out, and the same thing is true for Montana and Washington. Yet the United States Geological Survey indicates 44 billion tons of bituminous reserves.

Washington is credited with 1100 sq miles and 20 billion tons, indicating coal over 20 ft thick, while the 1936 figures of 63.6 billion tons would indicate coal 60 ft thick over the entire area, although the 1936 tonnage may have included a larger area.

In *Montana* over 34,000 sq miles, or

about ¼ the area of the state, is said to contain 303.1 billion tons, or a 10 ft bed of coal mostly lignite.

New Mexico is said to contain 163.8 billion tons spread over 13,000 sq miles representing coal 13 ft thick. The United States Geological Survey's report of 1936 reduced this to 20.7 billion tons, but the United States Coal Commission's report calls for 97.2 billion tons, while the average thickness of seams worked is 5 ft.

In *North Dakota* nearly half the area of the state is shown by the 1907 report to contain 500 billion tons, and by the 1936 report 600 billion tons of lignite, or equivalent to a 20 ft seam over that entire area, while the seven operating mines reporting show coal of 7 to 20 ft or an average thickness of 12 ft and a life expectancy of 20 to 30 yr.

In *Wyoming* over 20,000 sq miles are said to contain 424 billion tons, or the equivalent of a coal bed 22 ft thick over that entire area. The Green River field with 4855 sq miles and 199.2 billion tons would indicate 41 ft of coal over the whole area. Or, if the 1350 sq miles of "*possible*" coals were included, it would still indicate a deposit 33 ft thick. To say the least, these figures seem excessive.

The purpose of this paper is to emphasize the need for a prompt and realistic survey of our coal resources, showing the thickness and depth of the seams, the quality and character of the coal, and its location with reference to the needs of our whole economy.

It was asserted at the Denver meeting last October that no one should have the temerity to question the Government figures unless they submitted maps and records proving their statements. Well, that is quite a burden to impose upon an individual to justify an opinion regarding our coal reserves. But that is exactly what could be done in many cases in the east where many have knowledge of the wide discrepancy between the Government figures and private records based on prospecting and actual development. It is these wid differences that prompt the fears and lea to the belief that these fantastic and unbelievable figures of the United State Geological Survey are wrong and dangerously misleading and should be correcte promptly.

At the risk of severe criticism, because o my interest in the subject, I have the urg to give you my guess as to our coal reserves I would fix the total at over 223 billio tons, or sufficient at present productio rates and mining losses for 250 yr. But i the highly industrialized east, where 9 pct of the coal is produced and used, estimate over 73 billion tons, or only 9 yr supply.

The coal industry has been so destructively competitive over the years tha much coal has been lost in mining tha could have been recovered for a few extr cents per ton. Coal mines have been abandoned for economic reasons, resulting i the loss of millions of tons of coal, for mos of which the cost to recover would b prohibitive.

The limitation of our total coal reserve is serious enough, but of more concern i some Eastern fields is the serious decline i deep mine production which has been goin on for some years past. Were it not for th large and increasing strip coal tonnage, shortage would now be felt. From 20 t over 40 pct of total product in some Easter fields is strip mined coal, and about 20 p in the nation as a whole now comes fro stripping. In most areas strip mining confined to a narrow strip of coal aroun the outcrop of the seam which was n recovered in underground mining becaus of poor quality or bad roof condition Much of it could not be marketed now b cause of quality except for the strong d mand and the inability of consumers to g other, better coal. While there are som strip areas where underground mining ha not been done and overburden up to 85 an 90 ft is being handled, in most cases it done in shallow areas and stripping cannot be expected to provide a very large tonnage for more than 5 to 10 yr.

A perusal of Coal's "Blue Book" shows a surprisingly long list of mines, some large producers, with a life expectancy of 10 yr or less. Not all companies give this information, and those reporting are prone to show larger reserves than actual. Many mines not reporting are known to have reserves for less than 10 yr.

But the fact is a decline in deep mined production is already under way. In an Eastern area producing nearly 200 million tons annually it is predicted that the decline in production of deep mined and strip mined coal will be 90 million tons in 10 yr. Much of this production, *but not all*, can be made up by new openings in undeveloped coal. In many areas there is no virgin territory to replace worked out mines. The situation in the East in 20 yr will be immeasurably worse, as many more mines will be exhausted with no new areas available for development.

The growing shortage of low sulphur coking coals is not new to this group. Recent purchases of mines by the steel nterests have taken out of commercial production some large tonnages, with more to ollow. It may be surprising to learn that of the 7033 coal mines reporting in 1945, 140 mines produced 12½ million tons, or an average of only 15 tons per day. Another 920 mines produced 45 million tons or less han 90 tons per day. These are mostly amily or co-operative groups, mining the dregs, from little patches of crop coal left in by the larger abandoned operations.

The Bureau of Mines says, "When the bituminous coals are exhausted, particuarly in the East, and our natural oil and as is gone, we can then get liquid fuels rom the 'enormous proven deposits' of ub-bituminous coal and lignite located in he Dakotas, Montana, Wyoming, Coloado and Texas." If their estimates of these ow grade solid fuels are as erroneous and misleading as are their estimates of bituminous coal in the East and West, we may find ourselves paying dearly for the shocking waste now going on in the production and use of our valuable and fast disappearing natural resources: oil, gas and coal.

Our nation may become one of the "have nots" sooner than we deserve.

If we admit the existence of large reserves of subbituminous coal and lignite in the West, we should be informed as to the cost of replacing in small measure what we are now wasting in the east. As mined, lignite contains 28 to 45 pct moisture—something like a sponge for holding water—and subbituminous coal from 15 to 28 pct moisture. When exposed to air, the moisture evaporates and the coal disintegrates, breaking into small pieces. These fuels also have a bad habit of igniting spontaneously and can be stored safely only under water or in closed bins, so that they cannot be shipped and must be used or processed near where produced.

A few years ago, a Bureau of Mines' report indicated that to produce gasoline from coal by the hydrogenation process would cost 10 to 15 cts per gal more than from natural oil, and that the cost to produce gasoline by the Fischer-Tropsch process, *if future improvements contemplated are a success*, will be 6 to 8 cts per gal more than from oil. These figures need reviewing, as the cost to produce coal has doubled in the past few years, although with extreme scarcities cost becomes less a factor.

Imagine if you can in the not too distant future depending upon "bottled gas" from the Rocky Mountains or the Black Hills for energy for our highly developed economy in the east.

It is not alone oil and its products, particularly motor fuel, we need be concerned about; but fuel for power and heating, for industry, the utilities and every service now depending upon coal and oil.

We are hearing much about synthetic fuels, the commercial feasibility of plants to convert coal to oil and its products, gasoline and even gas. The cost to build a plant for 10,000 gal of gasoline daily is said to be $100,000,000.

To justify this large investment would require an area of coal 15,000 to 20,000 acres, and there are comparatively few such large undeveloped tracts of coal in the eastern area.

It is said nearly 50 pct of natural gas is used for industrial purposes in competition with coal and at prices much below the equivalent cost of coal. The Federal Power Commission has approved certificates of convenience and necessity for lines not yet constructed and is considering others which, if granted, will displace millions of tons of coal. This competition can only result in the abandonment of many high cost mines and the further loss of millions of tons of coal underground, every ton of which should be conserved.

This silly waste of natural gas, nature's purest fuel, should no longer be condoned.

In view of the tragic shortage of oil and the near shortage of coal, this final act of the gas interests seems like the last grand fling of an improvident people who should want rather to conserve natural gas for the comfort and convenience of the domestic consumer, for which use it is best adapted.

From my many years of experience professionally and as an interested traveler in the coal fields of the United States, of one thing I am convinced, and that is we should no longer think of our coal reserves in terms of aeons or even centuries, but they should rather be thought of and reviewed in terms of decades if we are to make proper provision for our expanding needs "and make the most of what we have."

No one should be more interested in the conservation of our fuel resources than the mining engineers of the AIME. It would be natural for our citizens and political leaders to look to such an organization for guidance. It is imperative that we learn the facts as to our coal reserves and that right early. It is hoped this paper will provoke the discussion and action necessary to get started on a survey, because, in my opinion, the very security of our nation is dependent upon it.

TABLE 1—*Estimates of Coal Reserves*

State	Area Sq. Mi.	U. S. Geological Survey Original Coals, Billion Tons		U. S. Coal Commission
		1907	1936	To Jan. 1 1945
Alabama	8430	68.9	66.6	32.7
Arizona	30	.1		*
Arkansas	1684	1.9	1.3	1.0
California	500	1.0		*
Colorado	10130	371.8	316.6	157.5
Georgia	167	.9	.9	*
Idaho and Oregon	430	1.6	.7	*
Illinois	35600	240.0	197.8	52.7
Indiana	6500	44.2	52.0	26.2
Iowa	12560	29.2	28.7	14.3
Kansas	3100	7.0	29.7	14.3
Kentucky	16670	104.0	121.5	61.2
Louisiana	8800			
Maryland	455	8.0	7.7	.3
Michigan	11000	12.0	1.9	.1
Mississippi	7500			
Missouri	16700	40.0	83.7	20.2
Montana	34067	303.1	381.0	183.2
New Mexico	13335	163.8	20.7	95.2
North Carolina	60	.2	.1	*
North Dakota	31240	500.0	600.0	*
Ohio	12660	86.0	92.0	34.1
Oklahoma	10000	79.3	54.8	18.2
Penna. A	480	21.0	14.6	
Penna. B	14200	112.6	103.3	32.3
South Dakota	2000	10.	1.0	*
Tennessee	5400	25.7	25.3	8.3
Texas	10200	31.0	30.9	14.1
Utah	13130	196.5	93.2	46.3
Virginia	1900	22.5	21.1	10.5
Washington	1100	20.0	63.6	32.1
West Virginia	17000	231.0	147.7	63.9
Wyoming	20568	424.1	620.3	340.4
Other States	*			356.0
	327,596	3157.2	3180.8	1614.8

* Includes Arizona, California, Georgia, Idaho, North Carolina, Oregon, North and South Dakota.

DISCUSSION

(*Henry F. Hebley and E. E. Finn presiding*)

G. H. CADY*—Mr. Crichton has performed an important public service by calling attention to the need of a revision of the inventory of the coal resources of the United States. Of greatest need is such a new inventory of the high-grade, low and medium volatile coking coals, the supply of which apparently is becoming alarmingly short.

It should be pointed out, however, that systematic areal mapping of our coal resources, which is the only adequate basis for resource inventories, except for a very few states, has reached a very low level of activity. Actually we are dependent largely upon areal geology mapping carried on many years ago and except for information which may be in the file of mining companies, and is difficult to assemble, little additional knowledge about the resources is available than when the last systematic inventory was made by the late M. R. Campbell. An accurate inventory of our coal resources will never be possible unless systematic investigations of coal resources are resumed on a scale commensurate with its importance. There is much danger that in the momentary enthusiasm for revision of the inventory that this will be attempted without the essential preliminary studies that will require several years, and with the result that for many parts of the country there will be little improvement in the estimates over those made by Dr. Campbell, with the possibility that they will be unreasonably reduced.

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It is also well to point out that the nature of a reserve varies from year to year as mining methods and distribution and utilization practices are modified. Coal resources which would have been regarded as commercially realizable reserves 40 years ago in the light of then existing mining conditions and practices, appear no longer to qualify in this category, judging from the recent shifts in mining districts. Although these "abandoned" coal supplies no longer appear to qualify as marketable reserves, is it justifiable to regard such coal as an entirely negligible potential energy resource?

The Illinois State Geological Survey has devoted much time and attention in the compiling of accurate inventories of the coal resources of the state. However, because the deep-lying coal beds in the north half of the Illinois coal field have not been regarded with favor for many years by the coal mining industry, exploration of the resources in that area stands about where it did nearly 40 yr ago. It is doubtful whether a much more accurate idea of these resources, except in the outlying strippable areas, would vary much from that which formed the basis of Dr. Campbell's estimates. The change in the inventory, if any were made, would result from adopting new bases of evaluation in the light of recent trends in the mining and utilization of coal. It is suspected that similar conditions exist in many coal fields. Hence, it is necessary to come to an understanding in regard to the bases of evaluation that will be adopted in preparing an inventory of coal resources as determined by present practices and reasonably probable future changes.

Furthermore, it should be realized that quantitative estimates are inescapably highly variable, for example, a change of one foot in the estimates of the average thickness of coal in the approximate 37,000 sq miles of the Illinois coal basin, would result in a change in the estimates in the order of 40 billion tons. Precision in the estimates of average thickness are exceedingly difficult to attain and cannot be attained in many districts without much more information than is yet available.

A. C. FIELDNER*—Mr. Crichton has presented a timely and important paper in view of present concern regarding our fuel reserves. We have accepted with comparatively little revision geologically inferred estimates made a number of years ago. It is my personal opinion that these would be scaled down, perhaps considerably, by a present day investigation, but not so drastically as suggested by Mr. Crichton. I am in complete agreement with him that an up-to-date survey of our coal reserves is of fundamental importance in the national interest. If our minable coal deposits are less than 10 pct of the commonly used estimate of three trillion tons, then we need to turn over a new leaf in our national attitude toward mining practices which recover only one-half to two-thirds of the coal in the deposits.

A field investigation, covering 10 states, of losses in mining bituminous coal was made by the U. S. Coal Commission and the Bureau of Mines in 1923.[1] In this study the losses were classified as "unavoidable or as avoidable without undue danger to life, without undue labor per ton of product, much in excess of that now prevailing in the bituminous mines of the country, and without serious damage to buildings and engineering works on the surface." The unavoidable losses were found to average 15.3 pct of the coal in the mines then operating and the avoidable losses 19.4. The total loss was 35 pct and the average recovery 65.

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* Chief, Fuels and Explosives Division, Bureau of Mines, U. S. Department of the Interior, Washington, D. C.

[1] George S. Rice and J. W. Paul: Amount and Nature of Losses in Mining Bituminous Coal in Eastern United States. Report of the U. S. Coal Commission (1925) part 3, 1841–1876.

These losses refer only to current operations of active mines and consist of: (1) Coal left on the roof and bottom. (2) Coal lost in room, entry, and panel pillars. (3) Coal lost in pillars around oil and gas wells. (4) Coal lost under buildings, railroads and in boundary pillars. (5) Coal lost in handling and preparation, underground and surface. (6) Coal lost in rolls, thin or dirty areas.

Rice, Fieldner, and Tryon[2] in 1935 called attention to additional losses and irrecoverable coal such as patches of uneconomically minable coal left behind when a mine is abandoned, and areas so small in extent or so isolated that even for the future they give no promise of being mined economically. They considered the total amount of such waste as indeterminate but undoubtedly large.

It is not known whether the greatly increased degree of mechanization of coal mines since 1923 has increased or decreased mining losses. We should know the facts and conduct a survey of the percentage of recovery in present day practice. This should include a breakdown and analysis of the losses and their causes, so that steps could be taken in solving the problems involved in obtaining a greater national recovery of coal. The following summary bears directly on some of the questions raised by Mr. Crichton:[3]

"The central problem in conservation of coal resources is not absolute exhaustion in the remote future but a relatively early increase in cost through depletion of the richer and more accessible deposits. The anthracite deposits of Pennsylvania are 29 per cent exhausted, and the anthracite industry has already entered the stage of increasing costs. In the bituminous fields the life of certain of the finest seams of the Appalachians is limited to 30 to 100 years at 1929 production. Resort to thinner or less accessible beds, of which large reserves are available, will increase the difficulties of mining, and under really difficult conditions the labor required per ton may be doubled or quadrupled. Avoidable wastes in mining bituminous coal range from 5 per cent to 30 per cent of the seams now worked, the average being 19.4 per cent. In an ordinary year the avoidable loss is 136,000,000 metric tons.

"Prevention of these wastes is less a technical than an economic and institutional problem. In many cases recovery of pillars may add slightly to the cost of production, and under level agricultural lands damage may result from surface subsidence. A small increase in total cost, however, would make possible a very large increase in recovery.

"Improvement is made difficult by the destructive competition characteristic of the industry, resulting from scattered ownership of the abundant resources, existence of 6,000 competing mines in 31 states, and a huge excess of capacity. As long as mining operates in surroundings of poverty, the individual operator can hardly avoid wasteful practices, and the individual states, which have constitutional authority to prohibit waste, are unable to act. The necessary first step in reducing waste therefore is economic stability within the industry and provision for collective action, which cannot be achieved without assistance from the Federal Government. If the industry can be established on a stable and profitable basis, the states may be able to exercise their constitutional powers by regulating wasteful practices. state legislation limiting the mine operator's liability for surface subsidence to the actual cost of restoration also is needed."

This quotation shows that the Bureau of Mines also has called attention to the importance of conserving our coal resources. Mr. Crichton has done well to call our attention to the need for revising estimates of coal reserves on the basis of up-to-date information. It is obvious that industry and the state and Federal governments should undertake such revision as soon as possible.

R. C. Moore* and E. K. Nixon*—Although we do not agree in detail with some of the conclusions of Mr. Crichton, feeling that he has cut the cloth a bit too narrow, it is nevertheless our belief that his ideas are in the right direction and that his giving the paper had the effect of waking us up a bit on the domestic coal reserve matter.

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[2] George S. Rice, A. C. Fieldner and F. G. Tryon: Conservation of Coal Resources. *Trans.* Third World Power Conference, Washington, D. C. (1936) **6**, 690–691.

[3] Pages 720–721 of reference 2.

* State Geologist and Geologist, respectively, State Geological Survey, University of Kansas, Lawrence, Kansas.

Probably it goes without saying that most informed engineers and geologists have taken the 3000-year coal reserve figure of the federal agencies with a grain of salt, but we have had nothing else more accurate to put forth.

We feel strongly that there is a sufficient volume of doubt on this coal reserve matter to justify the federal agencies making a new and thorough job of determining actually and conservatively what the facts are on proved, probable, and possible domestic coal reserves.

We might point out that the figure the author gave for Kansas' coal reserve, namely, 0.8 billion tons, was very close to the figure of "proved coal" estimated and published by our Survey within the last two years.

H. N. EAVENSON*—The timely and thought-provoking paper of Mr. Crichton will cause many headlines in the press, and probably some statements from those responsible for the data criticized, but it is a very much needed statement on the subject by one well versed in it.

This writer has not made a critical analysis of the accepted data such as the author has and will not attempt to establish the accuracy of either set of figures. Those in this paper represent the opinion of a mining engineer as to the reserve that can be recovered on an economical basis, while the geologist's figures represent estimates of *all* the coal in the ground, including some of which there is no real knowledge, and in many cases, little likelihood of existence.

Apparently the general public believes that this nation has ample coal reserves to provide fuel for some thousands of years, even assuming that the present users of natural gas and petroleum products will be compelled by their scarcity to turn to coal either as a direct fuel or as the source of the liquid fuel they would prefer to burn, and at a cost that will not be far different from what is now being paid, although, of course, they expect that recurrent strikes will continue to increase costs somewhat.

This belief is not in accord with the actual facts, and it is time the true condition is ascertained, and that our national policies be changed accordingly.

It is reasonably sure that our total reserves of coal will be much larger than Mr. Crichton states, but it will also be found that mining most of the additional amount will be much more costly than the present production, and in fact that long before the coal reserves are exhausted, it may be that some other source of heat energy may be found to be cheaper.

Funds should be provided to correlate all the existing information about our coal reserves, and to obtain enough more to enable a reasonably correct estimate to be made. This writer has supervised drilling in many Appalachian areas and has invariably found that coal seams of any importance are very scarce long before the 3000-ft depth limit is reached, and in the eastern fields, at least, he is very skeptical of any considerable quantity of even thin coal being found more than 1000 ft deep. Such an estimate, when made, should differentiate the reserves estimated between those capable of being mined under presently known methods and those whose existence is only problematical. The specifications of such reserves should properly be made by the committee recently appointed by the Institute to initiate such a program.

Mr. Crichton is to be congratulated on having spent so much time and energy on this subject and on his courage in expressing his opinions about it.

H. M. BANNERMAN*—Mr. Crichton was kind enough and courteous enough to send a copy of this paper to the Geological Survey several days ago. As a result, Dr. Wrather, the director, planned to attend this session for the purpose of participating in the discussion. However, he was unable to attend because of urgent matters relating to the AIME, and for that reason he asked me to offer this short comment in his behalf.

We are very much interested in Mr. Crichton's forthright opinions concerning the amount of coal available in the United States. Frankly, we think he has taken an extreme view. Parenthetically I might say that the Geological Survey's estimates may be at the other end of the extreme. However, the problem is obviously one which cannot be resolved by debate even though we are firmly convinced that the prospects for the future are not as gloomy as Mr. Crichton predicts.

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* Eavenson and Auchmuty, Pittsburgh, Pa.

* U. S. Geological Survey.

We agree, however, that the total reserves as published are probably too large. We agree that the thicker beds of high rank coal are being rapidly exhausted; that considerable reserves of subbituminous coal and lignite are both low in heat value and remote from centers of consumption. Nevertheless on a nation-wide basis we believe that the reserves of coal are comfortably large as compared with our current and probable annual needs. To be sure, much of the easy coal is gone.

We will have to work harder for that which we get in the future. We will be forced to mine thinner beds and beds less desirable and beds more remote from consumption centers. This is in keeping with the history of mining as a whole. In other commodities there has been a general trend in the grade of material that is considered minable, and it is only reasonable to assume that coal will follow this pattern. In our opinion, therefore, Mr. Crichton has done the coal industry and the public in general a very great service in pointing out the need for more and better information about our coal reserves.

But the problem of reappraising the nation's coal reserves is costly and time consuming. The only reliable basis for estimating coal reserves is detailed basic surveys coal field by coal field, mapping outcrops, measuring coal thicknesses, and studying the characteristics of the roof rock, thickness of overburden, and so on. The Geological Survey has been engaged in this task for many years and yet there are vast areas of our coal fields which are still unmapped and many others are inadequately known. The preparation of coal reserve figures by states or by fields represents primarily the summing up of detailed studies, and the totals have value and the users of coal will have dependable data, only if the estimates are accompanied by detailed tabulations by rank, by bed, by thickness, by township and by county.

The last significant revision of the summary reserve figures made by the Geological Survey were made in 1928. Since then we have been unable to carry on this work on an adequate scale partly because of lack of facilities, partly through pressure of more urgent work occasioned by World War II. Since the war, however, we have renewed our attack upon the problem. At the present we are carrying on detailed mapping in eight areas in the United States: Powder River, Mont.; Durango Field, Colo.; Jasonville, Ind.; Pennsylvania Anthracite, Deep River Coal Field, N. Carolina; Knife River, N. Dakota.

In addition, we are currently engaged in summarizing such data as are available on the total coal resources of Montana. We have definite and detailed plans for increasing the scale of our work on coal during the coming summer, but whether or not these plans will materialize depends primarily on whether our request for funds to support the work will be approved.

A. B. Crichton (author's reply)—The discussions of Dr. G. H. Cady of the Illinois Geological Survey, and Dr. A. C. Fieldner of the Bureau of Mines, bear too much upon the economics of coal mining. The amount of coal that can be commercially produced and marketed under modern mining, distribution and utilization methods is an important consideration, I admit, but it refers not so much to original reserves as to recoveries in mining. It leaves the inference that there are vast deposits somewhere in the ground that can be recovered as needed later but at a cost not now justified.

It is still my opinion that the original estimates of the United States Geological Survey and most State Geological Surveys are greatly overstated, that serious mistakes were made as to the coal bearing areas and as to the correlation of seams, and that a new and accurate survey is long overdue. The comment of Moore and Nixon of the Kansas Geological Survey would seem to prove it.

Mr. Eavenson's discussion is appreciated. He has had many years' experience in the coal fields of our country and knows the United States Geological Survey figures greatly overstate our coal reserves. He sees the need for an up-to-date and realistic survey at once.

The discussion of Dr. Bannerman of the United States Geological Survey shows how grudgingly they yield on coal reserves by his statement that the "total reserves are *probably* less than the published figures," while a cursory examination of private data readily available would show that there is probably only a small fraction of what published figures show. Why does the United States Geological Survey continue to quote the high Campbell figures when more recent State Geological Survey figures show so very much less?

He agrees that the thicker and higher rank coals are being rapidly exhausted and that much of the easily accessible coal is gone, leaving the inference there is hidden away in the hills enormous deposits in thin seams at great depths or coal of an inferior quality not now marketable. Where are these coal deposits? This is another guess without much basis of fact as there are few such deposits in the eastern fields that could in any substantial measure make up the large production now maintaining our economy, and which production from deep mines is now so seriously and rapidly declining. There is already being mined in some fields coal so thin and of such poor quality that the energy expended in its mining and preparation is almost equal to that obtained from the product itself, something comparable to our atomic bombs.

The work now being carried on by the Survey is mostly in the little developed fields in the far West while the area that needs most attention is the highly developed industrial area in the East, where 92 pct of our bituminous coal is produced and used. In my humble opinion the situation is far worse than my present paper indicated. About 25 pct of coal produced is now coming from shallow strip pits with a definitely limited life. In many of our largest producing mines the coal will be exhausted within ten years with comparatively few undeveloped areas to replace them. With the growing demand for coal in the East a shortage is likely in 10 yr and in 20 yr, unless the oil situation eases, the shortage of fuel in my opinion will be very serious.

Let it be understood the sole purpose of my present paper and this discussion is to provoke action. I am sure the present Bureau of Mines and United States Geological Survey officials would gladly proceed with a survey if funds were available for the purpose, and because I feel nothing is more important to the security of our country, the AIME and the Coal Industry should do all possible toward that end.

P. H. PRICE*—The estimates of original coal in West Virginia made by the State Geological Survey were generally computed from planimetric measurements of the acreages shown outcropping on the geological maps (1:62,500) of the various county surveys. On seams too close together to show both, the acreages of one were interpolated from the other. The thicknesses used were of necessity averages but were based upon actual seam measurements, core tests when available and borings for oil and gas.

Some of the recent stripping operations in the State show a considerably greater coal thickness than was used in our computations as the average. In some of the counties, Pleasants, Wood and Ritchie for examples, no estimates were included although drilling has indicated the presence of coal there. No estimates were included for the Pocono coals known to be present in Morgan, Pocahontas, Greenbrier, Mercer, Monroe and Summers counties.

More recent boring for gas and oil has proved the coal seams to extend beyond the boundaries used in the original computations.

While it is of course impossible to determine beforehand certain "wants" in coal seams it is believed that in general our estimates are on the conservative side. We know of no better way to arrive at the original tonnages and must therefore support our original estimates.

A. B. CRICHTON—There can be no criticism of Dr. Price or the method used by the West Virginia Geological Survey in their estimate of coal reserves. The West Virginia Geological Survey has probably done more field work and prepared more maps and reports on coal reserves than has any other state. But subsequent mine development and careful prospecting have shown there is much less coal in the principal producing fields than the State Survey figures indicate and this was pointed out in my original paper.

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* State Geologist, West Virginia Geological Survey, Morgantown, West Virginia.

Considerable core drilling has recently been done in Greenbrier and adjoining counties and no Pocono coals have been found of commercial value. The coal reserves in the seams now being worked in the Greenbrier area are very much less than the estimates of the State Geological Survey. After all, the important thing I have been trying to emphasize is the need for an up-to-date and realistic survey.

Symposium on Grouting

(New York Meeting, February 1948)

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Introduction—Grouting in Mines

BY F. C. STURGES,* MEMBER AIME

BY definition the word "grout" means a thin mortar, or a kind of plaster or cement, and "grouting" means to fill up or finish with grout. The words "cement," "plaster" and "mortar" mean a substance that is used in a soft state which subsequently hardens. Thus, grouting is the filling of void space with a substance that hardens. Construction cements are the most common substances used, either by themselves or in mixtures, but the term grouting is applied quite correctly when other substances such as plastics and certain chemicals are used. Also, the use of the term grouting to cover the filling of voids with materials that do not actually become hard, such as the flotation tailings used in grouting in the southeast Missouri lead district, should be considered correct, as some stiffening does take place and, anyway, it is a better descriptive term than any substitute.

Engineers are interested in grouting because some of the materials with which they must work contain voids, and certain conditions can be improved if these voids can be filled with something solid and permanent. Grouting is done to accomplish several purposes: 1. To stop the passage of fluids and gases, 2. To cement materials together to make them stronger, and 3. To prevent consolidation of materials by filling void space with something solid as a replacement for the air and water that could be forced out under load.

The construction industry uses grouting extensively for all three purposes. Grouting is used to stop water percolation under and around dams. It is used to control water flow in tunnel driving and in shaft sinking. It is used to improve foundation conditions under civil engineering structures by permanently filling void space and thereby reducing settlement, or by cementing broken rock together and increasing its supporting strength.

The oil and gas industry does a lot of grouting, primarily to stop the passage of gases and fluids. The search for oil at increasingly greater depths has meant greater investment per well, and the oil industry has spent large sums on development of improved drilling methods. This

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research has resulted in the development of better methods and better materials for pressure cementing work on wells, and we are indebted to their engineers because these new developments are also available for other grouting work.

Mining men who extract their minerals in a solid state rather than a liquid state have some of the same problems to meet that their brothers in the construction and oil industries have, because they are working with the same basic material—the crust of the earth. This crust is full of cracks and crevices, and the mining engineer has to solve problems arising from the fact that these crevices usually carry water and weaken the roof and walls of his mine. However, the mining industry as a whole has not made as extensive use of grouting to solve these problems as do the construction and oil industries. This is probably because most mining men have had the opinion that grouting would be too expensive to consider. When it comes to discussing cost of grouting, little can be said except that the cost depends on the particular job. For instance, our charges for grouting have ranged from as high as $5.00 per cu ft injected to as low as 50¢ per cu ft, including furnishing the materials. Each job is a special case, and it should be pointed out that where a grouting job has a high cost per cubic foot the total cost of the job is generally low, and vice versa. The job mentioned above where the cost was so high per cubic foot was done with just a few sacks of neat cement, whereas the low cost per cubic foot on the other extreme was done with a mixture that was only 10 pct Portland cement and where the rate of injection was over 7000 cu ft per day. Where cementing action is important neat cement is generally used, but where the strength of the grout filling the voids is not so important, substantial savings can be obtained by using a mixture of a substance such as rock flour with a smaller amount of Portland cement.

The following table shows different substances that have been used for grout:

Table 1—*Grout Materials*

1. Cements
    1. *Portland Cements*—Regular Construction Cements—Grouting properties vary somewhat between manufacturers.
    2. *High Early Strength Cements*—"High-Alumina" Cements—Grouting properties include (1) Better penetration of small crevices because of finer grinding, and (2) Time saving because of early developed strength.
    3. *Quick-setting Cements*—Portland cements with an accelerator such as calcium chloride mixed in by the manufacturer.
    4. *Gypsum Cements*—Made from calcium sulphate. Set and harden to full strength at approximately the same time, in about 2 hr.
    5. *Special Cements*—Manufactured for special uses, such as:
        - (1) Retarded setting cements developed for oil well use at great depth or at high temperature.
        - (2) Iron oxide cements for high specific gravity and resistance to salt water.
2. Admixtures to Cement
    1. *To Accelerate Setting Time*—Calcium chloride (most commonly used), sodium silicate, sodium hydroxide.
    2. *To Retard Setting Time*—Sodium tannate, gypsum, lime, sugar.
    3. *To Bridge Crevices*—Bentonite, sawdust, grains, cotton-seed hulls, asbestos fiber, cellulose fiber, mica flakes, sand. A number of these substances are manufactured and sold under various trade names.
    4. *For High Angle of Repose*—Same as 3.
    5. *To Increase Plasticity of Grout*—Finely ground bentonite.
    6. *To Reduce Grout Shrinkage*—Finely ground bentonite.
    7. *To Lower Cost of Grout where Strength is Not Important*—Rock flour (usually obtained as a byproduct from a limestone aggregate plant), clay, ground shale, any fine material such as flotation tailings. Raw cement and sand have also been used but present difficulties in handling with ordinary piston displacement pumps.
3. Plastics
    1. Bakelite type plastics (phenol-formaldehyde compounds), injected in a molten state and harden on cooling.
    2. Asphalt.
    3. New synthetic resins or resin cements that harden from chemical action.

    Plastics have important grouting properties, such as resistance to contamination and ability to penetrate materials with low permeability.
4. Chemicals

    Sodium silicate mixed with calcium chloride, aluminum sulphate, or any heavy metal salt, plus, usually, the addition of an acid to control the time of setting. Useful for grouting materials with low permeability (as low as 0.0001 cm per sec.)

In some mines grouting has proved to be an effective tool in improving mine operation, particularly in lowering pumping costs, eliminating ice accumulation in air intakes, and improving safety conditions. In other mines it would not be worth considering. In order to aid mining engineers in understanding the latest developments and possibilities of grouting, this program was designed to accumulate information for study. We are greatly indebted to representatives of the construction industry, the oil industry, mine operators and grouting contractors for the information they are about to present to you.

Use and Technique of Pressure Grouting in the Construction Industry

By V. L. Minear*

This paper presents some of the problems encountered and solved by the construction industry during recent years while pressure grouting the foundation rock of dams. Pressure grouting has become "Standard Operational Procedure" and consequently many hundreds of thousands of bags of cement have been used for that purpose. There is nothing weird, mystical, or even particularly difficult about the basic idea. Holes are drilled in the rock and grout is pumped into them. This procedure is used to shut off the flow of water under the dam and to reduce the uplift on the dam. Like most other engineering and construction problems, the fundamental principle is simplicity itself, but some of the "minor details" are rather complex. When faced with the problem of actually forcing the grout into the rock, or keeping it there once it has been injected, an engineer frequently has need for all of the ingenuity and initiative that he has at his disposal. It is proposed to discuss these minor details at length, even at the risk of boring the reader. The difference between success and failure in a grouting program can well hinge upon some such detail that may not be fully understood.

Pressure grouting is a method used to make good foundations from bad ones by plugging actual or potential passages within the rock through which water might escape and thereby diminish or even destroy the usefulness of the dam. In some special cases, pressure grouting has been used for the purpose of consolidating broken rock to increase its bearing power. Lacking this useful method, many of the dams constructed during recent years could not have been built. Most of the really excellent dam sites in this country have been utilized long ago. Consequently it becomes necessary to find methods of utilizing the less desirable sites. Pressure grouting plays a vital role in the methods developed.

Engineers have learned the hard way, that the term "solid rock" is a misnomer. Solid rock does not exist to any considerable extent in nature. Practically all rock contains imperfections such as shrinkage cracks, bedding planes, joints, solution channels, folds, shear zones, crush zones, and the like. All these imperfections are weaknesses through which ground water is likely to flow. The function of rock grouting is to rectify these imperfections and make the rock as nearly "solid" as practicable.

The location, direction, depth and number of grout holes to be drilled are important. Clearly, a hole must pierce fractures in order to serve a useful purpose, for grout cannot be forced into the rock itself. Were it possible to determine the pattern and extent of the cracks within the rock, it might be feasible to drill two or three strategically placed holes so as to make practicable the grouting of the whole integrated network of cracks from them. Such is not the case. While a study of the geology, surface indications, and drill cores is helpful in gaining an idea of subsurface conditions, in the final analysis, the seams must be located by trial and error methods. There can be no doubt that the closer the spacing of holes, the more probable it is that all major seams will be intersected. Moreover, a close spacing of holes is conducive to economy of grout in that a shorter path of grout travel is afforded.

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The ideal tool for drilling grout holes is the diamond drill. This tool produces a minimum of sludge and consequently a minimum tendency to choke the fine, hair-like cracks. However, this method of drilling is an expensive luxury when short holes are to be drilled under conditions where high grouting pressures can be used without danger of rock displacement. Percussion drilling is much cheaper and obstructions can be broken through by the injection of water under high pressure prior to the introduction of grout. This will be readily appreciated by anyone who has ever tried to calk a seam in rock from which water is leaking.

The equipment generally used in mixing and placing grout has been described elsewhere* but will be presented briefly for convenience. The grouting materials are mixed in either a specially designed grout mixer or an ordinary concrete mixer. In either case a suitable water meter should be used for measuring the mixing water since the measuring device with which most concrete mixers are equipped is not sufficiently flexible for use with grout.

Injection is by means of pneumatic displacement apparatus, by pump, or by gravity. Pumps are superior for placing neat cement grout but they are not suitable for use with grout containing sand or other coarse materials. Injectors handle such materials satisfactorily. Sometimes gravity alone is used in grouting shafts. Below is the weight of a column of neat cement grout, one inch square and one foot in height. From this table, the pressure that can be developed for any effective head may be computed:

TABLE I—*Showing Unit Weight of Various Mixes*

*w/c**	Unit Wt in Lb	*w/c**	Unit Wt in Lb
0.75	0.798	2.00	0.614
1.00	0.736	2.25	0.598
1.25	0.692	2.50	0.584
1.50	0.660	2.75	0.572
1.75	0.634	3.00	0.562

* *w/c* measured by volume.

Actual grout injection is a highly skilled trade in which experience is of vital importance. Over thirty years ago, Mr. Robert Ridgeway, M. Am. Soc. C.E., stated,* "Grouting is expensive, and to secure the best results at the least cost and in the quickest time, it is wise to employ only those experienced in such work. To start with an untrained force usually results in discouragement and sometimes the abandonment of the work. After the men are well broken in, they seem to develop an instinct for doing the work in the right way, in the shortest time and accomplish what before seemed impossible." The author concurs entirely with this statement. The success of any rock grouting program hinges upon forcing cement into the offending voids and keeping it there. In accomplishing this work, the importance of experienced judgment cannot be overstressed.

Experience indicates that the amount of cement which can be forced into a given hole depends upon: 1. The character of the formation penetrated. 2. The consistency of the grout. 3. The pressure applied. 4. The rate of injection. These factors are closely related as will be shown later.

The character of the formation penetrated by a drilled hole, largely but not entirely, fixes the amount of cement that can be forced into it. Clearly, more grout can be forced into a rock in which numerous cracks, fissures, caverns, and the like, are present, than can be forced into a similar rock which is free from these defects; yet much can be done to increase the cement consumption of rock containing such defects to its benefit. Most of such defects, in their natural state, are partially, if not entirely filled with gouge or other material which must be removed before grout can be injected. Cement grout cannot be forced into mud for any appreciable distance. Moreover, any considerable mixing of the cement with the mud results in a structurally weak material lacking strength to resist the thrust or erosive power of the water in the seam. Incomplete removal of the mud before grouting often results in the water subsequently washing out the remaining mud, thus opening up a new channel, thereby obliterating any benefit derived from the grouting.

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* See *Civil Engineering* Nov. 1936, p. 751 and *Jnl.* of the American Concrete Inst., April 1947, p. 921.

* *Trans.* A.S.C.E. (1919–20) 106.

Drilling operations complicate an already complicated condition. This is due to a tendency for the cuttings to lodge in and obstruct the smaller fissures, regardless of whether percussion or rotary drilling is used. This fine, granular material acts as a filter which will allow water to pass, but will deny passage to cement or other suspended matter. This action builds a filter cake and reinforces the obstruction. However, the injection of water at high velocity, preliminary to the introduction of grout tends to open small continuous channels through this material by carrying the cuttings and gouge from the finer to the larger passages lying at a greater distance from the hole. Some engineers question the use of water for this purpose, particularly where a relatively tight rock is being grouted. Their objection is based on the belief that water may lodge in pockets and seams and occupy space which would otherwise be filled with grout. On the other hand, this objection may be answered on the theory that, under high pressure, the water will be forced into the rock pores or to other openings lying at a distance from the hole. If there is no such escape for the water, there is no need for grout in that particular crevice. Furthermore, there can be little doubt that when grout, under pressure, comes into contact with dry rock, this rock absorbs a considerable amount of the mixing water. This results in the grout becoming thickened, losing its fluidity and plugging the seam prematurely. For these reasons, the initial operation in grouting should be the introduction of water into the hole at the highest pressure which local conditions will permit. This operation should continue as long as there is any noticeable "pick-up," or increase in the pump's speed. Should the pumping operation cause an appreciable return of water from an adjacent hole, small "slugs" of compressed air should be introduced with the water in order to increase its turbulence and consequently its cutting power. This operation should continue as long as the return water is muddy.

The selection of a proper water-cement ratio is one of the most difficult and important phases of rock grouting. Adaptation of the consistency of the mix to the constantly changing conditions within the rock constitutes much of the secret of success in grouting. Undoubtedly more holes have been lost by inexperienced men attempting to use a thicker grout than the formation would tolerate, than by any other cause. Conversely, the use of a mix containing more than the optimum water content results in work of needlessly low quality. There is no safe rule for the determination of the mix to be used other than that it should contain the lowest *w/c* ratio that can be injected successfully. The pump's behavior, the rate of cement consumption and the reaction to various changes of mix are valuable aids in arriving at a correct solution. These signs are not infallible however and the solution depends largely upon the experienced judgment of the operator. The initial injection in each hole should be thin enough to enable the engineer to "feel out" the hole without the danger of losing it. This thin grout should be pumped only long enough to estimate the hole's resistance and to enable the engineer to proceed intelligently with the adjustment of the *w/c* ratio. Since this adjustment consists of progressively reducing the water content to the minimum that the formation will tolerate, abnormally thin mixes do not give much information and tend to prolong the period of adjustment.

The ideal pressure to be used in placing the grout is the maximum pressure which can be applied without causing rock displacement. There is no inherent virtue in high pressure *per se;* the advantage lies in the fact that it makes possible the injection of higher quality grout through fewer drilled holes than is generally the case when low pressures are used. The flow of grout through the cracks and fissures within the rock is comparable to the flow of liquids through pipes. Frictional resistance is very high in the fine, tortuous, hair-like cracks and must be overcome by the application of pressure. It would seem that the higher the pumping pressure used, the greater would be the spread of the grout from a given hole and consequently the fewer the holes that would be required to grout a given area. This, however, is of secondary importance to the quality of the work done. It already has been stated that the thickest grout practicable should be used since the quality of the hardened grout is improved by the use of a low, original water content. The unit weight of thick grout may be almost twice that of a thin grout and consequently a correspondingly greater force is required to move it. When a needlessly low pressure is specified, a needlessly high *w/c* ratio is implied.

A close control of the rate of pumping is necessary. This is due to the tendency of the water to drop suspended cement particles unless a reasonably high velocity is maintained in the pump, delivery line, and rock fissures. When this occurs, not only is trouble experienced with the equipment, but the quality of the work is impaired by the loss of the coarser cement particles in the grout. These particles settle to the bottom of the horizontal reaches of the pipeline until the cross section is reduced to a point where the velocity is sufficient to retain all particles in suspension. A similar action probably takes place within the rock. Under these conditions, should there be an increase in the rate of pumping, these coarse granules would probably be picked up, thicken the grout and possibly plug the hole.

The foregoing discussion is largely academic and is intended to explain some of the relationships noticeable during grout injection. The discussion reveals three inter-related variables: applied pressure, water-cement ratio, and pump speed. A change in any one of these variables will be reflected in one or both of the others. Thus, decreasing the water-cement ratio is accompanied by an increase in the indicated pressure and a reduction in the pump speed. Similarly, a higher speed with constant *w/c* results in a higher gauge pressure. It is necessary therefore, to coordinate the three variables. This is accomplished in practice, by maintaining the pressure and speed nearly constant, by manipulating the *w/c* and, to a lesser degree, the throttle. An effort should be made to pump at the maximum allowable pressure at all times. In case it becomes necessary to race the pump in order to obtain this pressure, the grout should be thickened until a reasonable speed is reached. If the pump labors at the required pressure, the grout should be made thinner.

Another "minor detail" which frequently causes major difficulties, is retention of the grout within the rock into which it is being injected. When pressure grouting operations are properly conducted, the liquid grout finds and escapes from any existing opening in the same manner that air escapes from an automobile tire. Patching operations are much more difficult with grout leaks, and some form of surface treatment is usually necessary to prevent the escape of grout when pressure is applied. This treatment may range from simple calking operations, through so-called "stage-grouting" to concrete lining.

The ability to seal leaks quickly is of prime importance. Costs, due to excessive leakage, can be very heavy. Unless the grouting operator has sufficient information to enable him to effectively stop the flow, the grout slips away from him and he is powerless to prevent it. He finds himself confronted with a situation which requires the grout hole, costing perhaps several hundred dollars, to be abandoned or the pumping operation continued with much of the cement leaking to waste. Discontinuing pumping, even temporarily, is dangerous. This is particularly true during hot weather and it frequently results in the loss of the hole. In any event, the quality of the grout job is damaged due to the segregation of the cement particles which begins immediately after the flow is stopped. Abandonment of the hole often results in a recurrence of the same leak when adjacent holes are grouted, so that the net result may be the abandonment of several holes due to a single leak and an admittedly unfavorable condition ignored.

Since grout leaks can be expected at any time, it is almost mandatory to use adequate equipment, a trained crew and experienced supervision. When it becomes evident that a given method will not stop the leak encountered, sufficient materials should be available to permit various methods of calking in rapid succession.

The calking materials ordinarily used are oakum, lead wool, wooden wedges and flash-set cement. The cement is considered the most useful and versatile.

The best flash-set cement known to the author consists of a blend of 1 part Lumnite cement to 2½ parts of Portland cement. Properly combined and used, this blend has the faculty of taking its initial set in 5 min., its final set in 20 min and has a compressive strength of 700 psi in 30 min. It can be used in either its dry state or its liquid state. In either case, exact proportioning and an intimate blend is essential. If desired to use the dry blend for such purposes as applying patches to leaking surfaces, or for guniting wet areas, the proper amounts of the dry cements are mixed in a concrete mixer after which they are resacked for future use. If it is desired to use the blend as a grout for cutting off flowing water, grouting in pipe connections, or for stemming for blast holes, the flash-set grout should be made by combining Lumnite grout with Portland cement grout, stirring vigorously for not more than one minute and depositing the blend immediately. Not more than equal parts of water and cement by volume should be used and the Lumnite grout should be poured into the Portland grout.

An excellent description of the use of this material is contained in an unpublished report, dated Oct. 1, 1936, by F. A. Backman, on the subject of "Field Methods for Pressure Grouting of San Jacinto Tunnel (Metropolitan Water District of Los Angeles)" which is given in part below:

"The grouting of the battery station at Cabazon resulted in the cutting off of approximately 650 gpm between this point and a little east of there as measured by the hydrographer. The battery station is a drift running into the left wall about 400 ft east of the beginning of the Pioneer drift. The purpose is to maintain the locomotive repair, battery charging, and such, near the heading. Considerable water and bad ground requiring timber were found on advancing this drift. Its use as a repair shop necessitated grouting and guniting so the two operations were more or less combined. Consideration is here given to the methods used. Long holes were drilled to intercept the water courses as far from their opening into the drift as possible. As many holes were drilled as were necessary to take the water away from its original outlet. Pipes were placed in these holes and calked in with lead wool. The open seams in the rock were then patched with a quick setting mixture of Lumnite and Portland cement. This sealing of the open seams was not entirely successful because of the inaccessibility due to the timber and lagging, but was an aid in retaining the grout in the rock. Additional holes were drilled into the seams near their original outlet, and into these holes a thick mixture of sawdust, aqua-gel and cement was pumped until it began to flow out of the pipes then discharging the water. This effectively sealed all but the finest crevices and permitted pumping at high pressure when the final grouting was done. Before attempting to grout the pipes containing the heavy flows of water it was deemed advisable to gunite the more shattered areas to give additional strength to the ground before applying the grouting pressure necessary to secure a satisfactory job. A certain amount of preparation was necessary before gunite could be placed as there were numerous small leaks that had to be piped before gunite would adhere to the rock. Short holes were drilled and ¾-in. nipples inserted to relieve the pressure while guniting. One part Lumnite and three parts Portland cement were mixed with dry sand (ratio 1 cement to 3 sand) and shot in the usual manner. The three to one Portland-Lumnite mix hardens in less than 5 min and permits guniting in comparatively wet areas that with ordinary cement would be impossible. After guniting, the ¾-in. pipes were grouted with cement grout placed by air through a ¾-in. hose using approximately 100 lb pressure. The larger flows were grouted with the use of Gardner-Denver pumps and were pumped to refusal at 1000 lb pressure. Most of the lagging was then removed from the timber sections and the entire area gunited to a thickness of from 1 to 5 in.

Mixes for Use in Cement Gun Only

Lumnite	Portland	Lime	Sand	Temp Water °F	Set	High Strength
½...	½	0	3	90	1 min	2 hr
½...	½	0	3	65	10 min	5 hr
1 cf..	0	5 lb	3	90	1 min	20 min
1 cf..	0	5 lb	3	65	10 min	2 hr

These are approximate mixes and will vary slightly with temperature conditions especially that of the aggregates."

Successful grouting consists of devising a technique which will satisfy requirements of the local geology, conditions, and the like. There is given below a bibliography listing articles in which are described various problems solved by the construction industry with pressure grouting methods. The interested reader is referred to these articles for further data on the subject.

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Technique of Pressure Cementing in the Petroleum, Mining, and Construction Industries

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Part I

In the petroleum industry, the process known as oil well cementing is the equivalent of pressure grouting in the mining and construction industries. The science of oil well cementing has been known and practiced for many years. At the present time it has been brought to a high degree of perfection in the handling of many different types of jobs, all of which are known as oil well cementing. In order to understand more easily these processes, we will discuss briefly several different types of oil well cementing jobs, the purposes for which they are performed, and the methods of their accomplishment. It is believed that some of these procedures, in a modified form, can be applied to pressure grouting projects in general.

Originally, oil well cementing consisted of placing a neat cement slurry about a steel casing set in a bore hole. Cement is used about these casings to protect them from the corrosive effects of ground waters, to protect the oil and gas zones from encroachment of undesired water from other levels, to prevent the blowing out of high pressure oil and gas zones from shallow levels, and to strengthen generally and support the casing being placed in the bore hole.

After the bore hole is completed to the desired depth, casing is run into the well, pumps are attached to this casing and circulation of the mud fluid in the hole is established; this circulation being from the inside of the casing into the annulus about the casing, and thence back to the surface. Next, a cement slurry is mixed and pumped into the inside of the casing string. A plug, which acts as a piston within the casing, is placed above the cement slurry. This is to separate the top of the cement slurry from mud or water which is used to shove the cement down through the casing and out into the annulus. This plug further serves the purpose of shutting off against a baffle in the bottom end of the casing string, thus stopping the flow of cement out of the casing and leaving the shoe, or lower end of the casing, surrounded by cement and also filling the annulus there above. Casing strings cemented in the above described manner may vary in length from a few feet of large diameter pipe to very long lengths of smaller pipe. As of the present date the longest string of casing which has yet been cemented is some 16,600 ft deep. As a general rule, the deeper the well and the more cement slurry used, the higher the pressures which are required to place the slurry. In parts of West Texas, it is not uncommon to use 4000 sacks to cement a single string of casing in place.

Machinery used to mix the cement and place it into the casing is mounted on heavy duty, powerfully equipped trucks which are capable of negotiating any type of highway or country roads. These trucks are equipped with two or more pumps which may be either power driven from the truck engine itself and auxiliary engines, or, where steam is available, the pumps may be of the steam type and use steam from the rig boilers. There are many types of these cementing units, ranging from very small sizes for use in areas where small amounts of cement and low pressures are encountered, to very large units equipped with twin diesel engines, and capable of producing pressures of more than 10,000 psi.

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The slurry is usually mixed by a hydraulic jet mixer which is composed of a conical funnel ending in a bowl shaped like a pipe elbow. In the back side of this bowl is located a small jet through which water is pumped at fairly high pressure. This jet action produces a vacuum in the funnel drawing the cement down into the stream of water and mixing it very thoroughly and rapidly by agitation in a short discharge pipe from the elbow. This cement slurry is then collected in a small vat where it is picked up by the suction of the pump which places it into the well.

Primary cementing jobs, as described above, have grown into many different forms, some of which may be applicable to grouting in a manner that the primary job is not. The principle type among these is known as "squeeze cementing." By this procedure, cement slurry is pumped down into a well through a small string of tubing or drill pipe and forced out into various zones in the well which are carrying undesirable water or gas. It has been established that cement, under high pressures, will penetrate into sand zones, probably by producing crevices therein, along the bedding planes, and in so doing will shut off undesirable water or gas. Many jobs of this type are done every day and are attended by a high degree of success in cutting off these undesired fluids.

In squeeze cementing there must be some pressure applied to the well bore to cause flow outwardly into the formation to be cemented. Filtration of the slurry by the permeable formation with which it is in contact causes the water in the slurry to pass out into the permeable medium leaving in place in the hole and in crevices, a very dense mass of cement in the form of a filter cake against the face of the permeable zone. It has further been found that if sufficient pressure is applied, the dehydrated slurry will obtain very quickly a high degree of strength, and will remain in place and not move on the release of the pressure applied to the zone.

These differential pressures in wells frequently run very high, as much as 9000 psi pump pressure has been applied; however, in shallow holes, and in highly porous and permeable zones, quite low pressures may be sufficient to allow for injection of large amounts of slurry. In any case some dehydration of the slurry toward the conclusion of the injection process is desirable and to accomplish this, a pressure higher than that required at the beginning of injection should be obtained.

The equipment necessary for oil well cementing is relatively simple to those who are experienced with it. It has been brought to a high degree of perfection and can be built along varied lines to fit any operation which might be necessary. It is entirely possible to produce equipment such as that mounted on the heavy trucks used in the oil field in smaller, more compact forms which could be lowered into a mine shaft, or to be used in tunnels. These pumping units could be equipped with various types of power, for example, electric motors instead of the diesel engines commonly used in the oil industry.

The hydraulic jet mixer is the least complex, speediest mixer available for the handling of large volumes of cement as are required in oil well cementing. This mixer is capable in one of its forms of mixing approximately 50 sacks of cement per min, and in smaller forms can be modified to mix a maximum of 15 to 20 sacks per min. This mixer can further be modified to produce cement slurries with varying water-cement ratios, particularly those of very high water-cement ratio, as are needed in most grouting procedures.

Numerous materials are used in oil well cementing, by far the most common of which is ordinary Portland cement mixed only with water to form neat cement slurries of varying water-cement ratios. For many years no other form of cement was available but as specific problems arose other cements and materials were developed to meet the required conditions.

High early-strength cements were introduced to save time in waiting for the cement to set up in a satisfactory strength. These cements are ground more finely than regular Portland cement and in some cases also have chemical differences. Ideally, they are characterized by pumpability periods about equal to those for Portland in the lower temperature ranges but acquire their final setting periods much more quickly than Portland and attain a strength in 24 hr about equivalent to Portland over a 48–72 hr period. These cements are essentially time savers which is a matter of considerable economic consequence.

As well depths become greater the time for cement placement and the temperature to which the cement is exposed increase rapidly. This caused introduction of the retarded or slow setting cements. A cement of this type is nearly always ground somewhat coarser than Portland and has mixed into it small percentages of various groups of chemicals which assist in slowing down the initial setting time. While the initial setting time is retarded and the pumpability period greatly increased at any given temperature, the final setting time for such cements is not materially changed from that of regular Portland and further the 72-hr strength of both types is about the same. Use of retarded-set cements is indicated in most instances where the temperature to which they are exposed exceeds 180°F.

Where highly mineralized ground waters are encountered, which are high in sulphates, most cements offer only limited protection as sulphates affect them rapidly. To combat these waters, special sulphate resistant cements have been developed and in some areas are widely used.

In most cases, regardless of the type of cement used, the mixed slurry is only cement and water, yet frequently various admixes are used in the slurry for several reasons. Very finely ground bentonite in small percentage is added to increase slurry viscosity, to prevent settling and to allow the production of light weight slurry with high water-cement ratio. Chopped cellophane flakes and certain other materials may be added to a slurry to cause bridging over of crevices and large porous areas which would otherwise rob the slurry away from the bore hole or cause loss of returns during the cementing process. To produce an inexpensive, very rapid hardening cement, calcium chloride may be added to the slurry; however in the use of this material caution is required, especially if temperature is in excess of normal atmospheric conditions. In cases where radioactive surveys are to be made to determine the final position of the cements, small amounts of carnotite may be added to the slurry.

Recently, there has been considerable emphasis on still further improved cements for oil well cementing. Much work has been done on a material known as resin cement, which basically is ordinary cement mixed with a water soluble synthetic resin. This particular material has not yet been completely developed, but has already shown a high degree of promise for squeeze cement work. When this mixture contacts a permeable zone, the liquid phase is squeezed out into the sand body in such manner that the liquid plastic will harden in place and seal off between sand grains. It can be proved in the laboratory that it is often impossible to force neat cement slurry into the interstices of a sandstone, because the cement particles are as large or nearly as large as the interstices themselves. However, if the pores of the sand are of sufficient size, it is frequently possible, with a high water-cement ratio to wash out into the sand enough cement to produce a sealing action. In the case of resin-cement slurries, the liquid resin and water phase easily penetrates the sand body without producing an early filter cake of cement and will harden in place while the remaining solids resembling the regular cement will harden in place in the larger openings and against the faces of the permeable zones. This material is of a critical nature with regard to temperature and must be balanced before use for the temperature of the zone in which it is to be placed. At the present time, these materials are quite expensive, costing approximately $20.00 per cu ft of slurry used. It is believed, however, that the cost of these special products can be reduced as supplies of the proper material become more readily available.

In some instances a synthetic liquid resin may be used without being mixed with the cement or any other filler. This is particularly applicable in dense sandstone where it is difficult to seal off the undesired water or gas by the mere cementing of the surface of the zone. In this case, the true fluid is easily placed back into the interstices of the sand. A liquid resin without admixes may be pumped great distances into the sand zone, and will disseminate through the sand in all directions. Therefore, sometimes control is necessary to prevent the material getting too far away from the bore hole or to prevent it seeking the more permeable zones rather than to go into all zones equally. A method has been worked out and an admix produced for liquid resins, so that they now may be placed across zones of varying permeability and yet insure an adequate flow into all sections of the zone regardless of its permeability. It is possible that the use of both the straight resins and the resin cement combination may have considerable application to grouting problems.

Another material which is frequently used in oil wells, is called Hydromite. This is a gypsum base material, similar to plaster of paris, to which has been added a dry powdered synthetic resin. Since Hydromite is capable of setting under continued agitation, and also can be controlled to solidify at various setting rates, such material is advantageous in the cutting of a fast flow of water.

Similar to Hydromite but not containing the resin admixture is Calseal, a highly refined gypsum cement. It is used in much the same manner as Hydromite but must be regarded as a temporary material since it is rather rapidly deteriorated by various ground waters. Calseal is widely used for the tamping of nitroglycerin and other explosives in oil well shooting.

PART 2

As mentioned earlier in this paper, many oil well cementing techniques are adaptable to pressure grouting. Likewise much of the equipment can be used for this work either as now built or with some modification. Regular materials used in oil well cementing are also the same as those used in pressure grouting and it is probable that new and specialized materials being developed may find a place in the latter type of service.

A large percentage of pressure grouting projects are remedial in nature. Some typical examples of this type of work are:

Dam Repair.
Building foundation stabilization.
Mud jacking concrete slabs and roads.
Mine water control.

Frequently a thorough study is not made of the foundation upon which a dam is to be built. This is particularly true of smaller municipal and privately owned projects. As a result, serious leakage may occur either under the dam, around the ends of the dam, or through the structure itself. Pressure grouting has been very successful in correcting these conditions.

If the fissures are small, a high water-cement ratio, neat cement slurry is injected. As mentioned in Part 1 of this paper, only a slurry of this type will penetrate such cracks and crevices probably because of a wide dispersion of the cement particles themselves in the slurry. Such dispersion reduces the bridging effect of the grains at the crevice opening, allowing them to enter in sufficient quantity to finally form a dehydrated mass that will withstand the injection pressure and give the desired sealing effect.

As the size of the opening increases, the water-cement ratio is decreased and admixes are added. Bentonite may be added to increase the viscosity of the slurry. Should a rapid set be desired, calcium chloride may be added.

If the cavities are large enough to require bridging before the slurry will remain in place, such fibrous material as cottonseed hulls or shredded cellophane may be added. In extreme cases, it may be necessary to gravel pack the cavities.

The use of various admixes must be tempered with good judgment. No specific rule can be set up as to type or quantity that is required since local conditions are the governing factors. However, it is important that such admixes do not retain the slurry to the area immediately adjacent to the injection hole only. This point is the major difference between oil well squeezing and most pressure grouting projects. In the former case, slurries are usually injected to protect the well, while in the latter case, the hole acts only as a point of entry to the formation.

A special gravel packing technique was developed on a dam repair job at Council Grove, Kansas, in June 1947. On this job, water was flowing under the dam through a limestone formation in which was found numerous mud seams and large cavities. There was a head of 50 ft on this formation with a very high water velocity. Heavy slurry with fibrous admixes was only partially successful and an excessive grout loss was experienced. One key hole was chosen with a 3-ft cavity. Very coarse sand and fine gravel were washed into the hole through the grout pipe, first with water and later with cement slurry. A total of 89 yd of this material was lubricated into the hole, water flow was substantially reduced and eventual refusal at 60 psi pump pressure was obtained.

Under certain conditions, building foundations may be stabilized by the injection of cement slurry. This process is especially adaptable to structures built on unconsolidated sands and gravels fully saturated with ground water. The technique may require the setting of grout pipes through the entire unconsolidated section, gun perforated at the proper levels and the slurry squeezed out into the sand, or it may involve the driving of the pipes only partially through the section, open ended, and forcing the grout out the open end upward and around the pipe allowing it to seek the more permeable zones. A combination of the two methods may also be used. In either case, the problem is to force cement outward from the point of injection through the more highly permeable zones in horizontal layers segregating the finer and less permeable sections, thus reducing either vibration or settlement.

If vibration set up within the structure extends downward into the sand, there always appears to be a reclassification of the material into fine and coarse layers. This reclassification becomes progressively less, the deeper the sands are penetrated and usually becomes negligible at a depth of 3 to 4 ft. Thus, the forming of horizontal layers of set cement, a short distance below the mat upon which moving machinery has been mounted, will stop the vertical movement of the sand grains and tend to reduce both vibration and settlement.

The greater the mass of cement injected, the greater is the assurance of success of the project. Therefore, some experimenting must be done with water-cement ratios to determine the proper proportions. Except in cases of very coarse sand and gravel, water-cement ratios of 20 gal per sack or more are normal. One case on record, namely The Kansas-Nebraska Gas Co. station at Deerfield, Kansas, required slurries with a minimum ratio of 30 gal per sack and at times it was necessary to increase this ratio to as high as 75–80 gal per sack. Lower ratios would immediately seal the exposed face of the sand preventing the injection of any slurry.

Since the slurry injected into unconsolidated material normally assumes a lenticular form in the more permeable zones, this type of work does not lend itself to the building of a grout curtain around a predetermined area unless the material is quite coarse and uniform throughout the section. It is probable, however, that a curtain could be accomplished by the injection of a material other than cement slurry. The use of synthetic resins has been considered but as mentioned earlier in this paper, the cost of the material usually is prohibitive.

The technique of gun-perforating grout pipes was especially developed for a foundation job performed for the Cities Service Gas Co. in 1941 at their Wichita, Kansas, pump station. In using this technique, a small gun, similar to the one used in deep wells, is lowered into the grout hole and fired at predetermined levels, perforating the pipe with either ⅜- or ½-in. steel bullets. Since that time, it has been successfully used on numerous projects. The number of perforations and their spacing depend upon local conditions.

Floors or slabs, poured directly on the ground, frequently sink below grade due to settlement of the underlying material. This condition may be corrected by forcing a slurry between the slab and the surface upon which it lies. This type of work requires very low injection pressures and care must be exercised so that the slab is not lifted above the desired grade line. Whenever possible it has been found advantageous to carry out the lifting operations in stages, thus reducing the stress on the slab. It is also important that the voids created be entirely filled with the injected slurry.

In the field of mining engineering, the problems of shaft sinking are comparable to those encountered by the petroleum engineer in the drilling of deep wells. One of the major problems in both instances is water control. This problem may be encountered either while the work is in progress or after completion.

Water intrusions after completion have been reduced or completely stopped by injecting cement grout through holes in the shaft walls or through grout holes drilled around the outside of the shaft. The first method is usually more effective if the walls have sufficient strength to withstand the necessary injection pressure. However, the physical difficulties encountered in making the set up often make it undesirable.

In some areas it is known from previous experiences that serious water difficulties will be encountered while sinking a shaft. Pre-grouting the shaft area will usually control this water sufficiently to allow the work to progress without expensive delays. Normally, grout pipes are set above the first water zone and a stage grouting program is carried out until each zone has been properly sealed. It also seems probable that the grout pipes could be set through all the waterbearing zones and selectively gun-perforated for squeezing. However, to our knowledge, this procedure has never been used.

The above methods of pre-grouting mine shafts are effective in hard rock areas only.

Attempts to pre-grout quicksand before shaft sinking have proved unsuccessful. It is believed that the use of straight resins or resin-cements might solve the problem in such areas if the additional costs of material could be justified.

Pressure grouting has also been called upon to do various miscellaneous remedial jobs ranging all the way from plugging old conduits to filling rat holes under grain elevators.

It appears that perhaps the greatest advantage the mining and construction industries can obtain from the oil industry in their grouting problems, is in the use of the special materials described, together with the multiplicity of tools in the way of packers, gun perforators, hole calipers, cement mixers and pumps which have been developed for use in oil well cementing and other phases of the petroleum industry. It seems likely that these devices can be applied to certain phases of grouting, especially so in both pre-grouting and remedial work on mine shafts and dams.

An Example of Controlled Pre-grouting in Shaft Sinking

By R. H. Allen* and J. W. Galpin,† Members AIME

Controlled pre-grouting is a technique developed during a period of more than ten years experience, in an effort to produce safer, drier and more economical mine shaft sinking. The technique involves a knowledge of hydro-mechanics, both static and dynamic, and a series of specially formulated grout mixtures to obtain several desired results through precontrolled setting time.

In general, the following procedures have been used to secure the best results in controlled pre-grouting:

1. Before drilling is commenced, one of two decisions is made: one procedure, a modified form of the Francois method, is to stage drill and grout, that is, drill a limited portion of the hole, then grout and redrill that section which has been grouted, progressing in stages until the required depth is reached. We prefer this method because no casing or grouting packers are required in the bore hole. This eliminates the risk of losing them in the bore hole due to packer failure, which might result in the expense of drilling a new grout hole. Another feature of this approach is that smaller drill holes may be effectively used, further reducing the costs. One drawback, however, is that because of the open and small size hole, it is impractical to attempt water draw-down tests.

The second alternative is to drill the bore hole, (generally NX size), to the required depth. The casing and grouting packer are placed in the bore hole at the lowest packer setting. A small, portable, power driven, swabbing device is used in the grouting casing to test the packer and remove mud or cuttings from the bore hole below the packer. The hydrostatic level is recorded and a close estimate or test is made of the quantity of water at each packer setting. Grouting progresses upward in a series of packer settings to the surface. Occasionally, casing is inserted in the bottom of the bore hole from within the mine, and water pressure and volume can be obtained in this manner. This is usually dangerous because of the unknown quantity of water.

2. The grouting, mining and geological engineers at the drill location make a detailed examination of the cores and also locate proper and firm grouting packer locations. A competent driller's opinion should be given due consideration as to the character and condition of the bore hole.

3. Several samples of formation water

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are taken from the bore hole at various depths. A pH test is made of each sample, and if necessary, a chemical analysis. Should any of these indicate sufficient chemicals to damage cement or its setting time, then preparation is made for storage and transportation of neutral water, or in some instances, special cements are ordered.

4. While the above hydraulic or drawdown tests are being made, the grouting engineer gathers several dry samples of the various cements and other grouting materials on hand which he expects to use within twenty-four hours. The samples are prepared with the mixing water to be used to obtain several different controlled setting times. Emphasis should be placed upon reproducing the approximate formation temperature and hydraulic pressure range of the sections to be control grouted. It is generally true that cold water retards and warm water accelerates the setting time of Portland, gypsum and aluminous cementing materials. Particularly is this important in controlled grouting. During the processing of these tests, others are made as follows: weight per gallon of grout mix (water-cement ratio), color, as some samples are made with various tracer dyes, viscosity, and the specific gravity. The tests are carefully recorded, and each is important in the technique of controlled grouting.

5. The last step before proceeding to grout is to pump water into the grouting packer point below the packer to record the hydraulic pump pressure at various injection rates in gallons per minute. A high pressure water meter is used in this instance.

6. I have selected, for this paper, an example where the use of conventional grouting methods indicated failure and was followed by the successful application of controlled grouting.

Location: West Virginia.

Proposed shaft: 18 × 18 × 500 ft vertical.

Proposed drilling pattern: NX size—9 holes—1 in the center and 8 in two concentric circles around the shaft site. No. 1 SE bore hole was grouted first on the proposed shaft line. The general character of the formations are gravel, sandstone, broken fissured shales and slates, interbedded with sand streaks. The formations usually contain subsurface water flows with interformational movement of fluids.

The depth of No. 1 SE bore hole is 517 ft drilled without any interruption for grouting. The casing and packer were run and expanded to the walls at 435 ft. Water was pumped into the formation to determine the operating pressure and found to take water at 22 gpm with 345 psi. This was followed by 155 bags of cement of unknown weights and viscosities, and with the use of no additives to increase the viscosity nor accelerators to hasten or control the set. The same procedure was used in the entire grout hole at five different packer settings. 1472 sacks of cement were pumped into the No. 1 SE bore hole with a maximum static pump pressure of 385 psi.

The drill machine was moved 27 ft to No. 2 NW bore hole and drilled to 485 ft. No cement grout was found in any of the cores recovered from the grout hole. The bore hole was grouted in the same manner, and with the same corresponding packer settings as No. 1 SE. The pump pressure variations, at the same grout injection rates, were unimportant. 1187 sacks of cement were used in this grout hole. The total number of sacks of cement in the two bore holes was 2659. It was then decided to redrill No. 2 NW and test for results. The test was made from within the mine, and indicated the bore hole to be making more than 250 gpm of water. The bottom of the bore hole was then plugged off and grout control started with the same five packer points. Five specially formulated batches of controlled flash setting, expansible grout were prepared with various tracer dyes—each batch being of 50 sacks, a total of 250. In conjunction with this, during the five grouting stages, 600 bags of cement grout with controlled viscosities and weights were also used. To fit individual grouting conditions, various angles of repose from 12 to 42° were necessary. The normal angle of repose of mixed neat Portland cement is 5° with no additives. Bentonitic colloidal clays were used to increase the viscosity with a resultant higher angle of repose. Bore hole No. 2 NW was regrouted completely with controlled grouting from bottom to the surface in 14 hr with 850 sacks of controlled grout. The bore hole was redrilled again and tested with 925 psi static pump pressure, and the dry test indicated 3 gpm of water in the bore hole, resulting in a decrease of 247 gpm. In this instance, interesting comparisons can be made of the two methods and the advantages of controlled pre-grouting over conventional methods.

No. 1 SE bore hole was again redrilled and seven grout control colored tracers were identified from No. 2 NW. Only 67 pct of No. 1 SE bore hole contained grout which had been previously placed in the conventional manner, and a large portion of it indicated pump erosion. No. 1 SE bore hole was tested and found to be making 97 gpm of water, and this quantity would no doubt have been larger if colored controlled grout tracers had not penetrated the grout hole from No. 2 NW. The bore hole was regrouted with controlled grouting, and comparable packer settings were again used. The operation proceeded in five stages. In each, 50 sacks of special, tailor-made, flash-setting, pre-controlled grout were used, followed by 100 sacks of controlled grout, in which the gravity and viscosity were held within narrow limits to suit individual stage grouting conditions. The control of this operation requires not only a thorough knowledge of the materials used and their behavior, but also, and we cannot stress this too strongly, experience in placing and setting. A total of 750 sacks were placed in the grouted area adjacent to this bore hole. The hole was redrilled following controlled grouting, and tested with the grouting pump at 1000 static psi. The dry test indicated 9 gpm, an 88 gpm reduction.

The drilling and pre-grouting were finished and all results were weighed. The company drilled only 6 grout holes, compared to an established program of 9—a saving of 3 bore holes or $4000.00 to $5000.00. Based upon their previous experience, 10,000 sacks of cement were appropriated for the job while only 6431 were used in controlled grouting; an overall saving in materials and drill time of $8000.00 to $10,000.00. The shaft is now completed and is making 8 gpm of water with one short water ring, the result of heavy blasting. With this exception in the sinking of the shaft, all walls stood firm as reported by the shaft sinking contractor.

Solidifying Mines and Shafts Areas by Pressure Grouting

By B. H. Mott,* Member AIME

Underground water has been one of the greatest problems in sinking mine shafts, sealing existing shafts, and driving headings under streams. In the preparation of a proposed shaft or existing shafts for elimination of water leakage, advances have been brought about through the use of a controlled grouting method.

The modern method to divert water from proposed shaft sites is known to us as pre-grouting, and from existing shafts, as controlled pressure grouting. This method involves the solidification of the surface and rock by the injection of a liquid mixture of cement and a certain percentage of rock dust and special clays pumped through a duplex plunger type slush pump that all cavities, fissures, and the entire rock structure in the area be made watertight.

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A three-fold advantage is readily found in employing such principles. 1. The elimination of the greater part, or all of the water problem in sinking a new shaft. 2. Sealing-off of leaks in existing shafts without interruption of mine operation. 3. Solidifying of wet mine entries as may be found in driving wet headings under streams.

In the sinking of a shaft, underground water is often a very costly part of its completion. Also, it has been necessary to abandon shafts because of water and unstable ground. Many methods have been devised to combat the difficulties in taking care of a large volume of water under pressure. This problem of sealing off water has not been completely solved, in spite of the numerous plans that have been tried. The freezing method has been found too expensive; as is also true with chemicals. The grouting method has proved the most satisfactory from an economy standpoint.

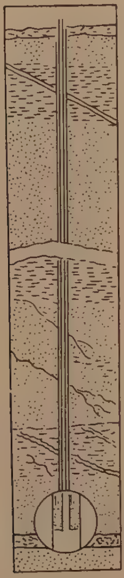
The filling of cavities and fissures in broken rock formations is not new; however, the old method of grout and grouting has ceased to mean merely a cement mixture pumped into the top of a drill hole and calling it grouted.

It may prove interesting to bring out some experiences in pressure grouting for proposed mine shafts and underground entries. One of the fundamentals is to predetermine at what elevation in the drilled hole grout would be accepted, and at what pressures. In addition, it is necessary to know the volume of water that fissures and cavities would take on a water test. This is done by employing a hydraulic sectional water-testing device that will segregate and test each 5-ft section of the entire hole for water leakage.

Such information is then carefully checked with the drilling log and core itself. The water losses, the inflow, openings, and soft ground, determine the procedure of grouting for that particular hole. If the test shows a large amount of water, rock dust or other filler will be used. As an example, there is used the seven-hole layout for a shaft (Fig 1). This number of holes usually has been found to be sufficient unless the shaft is adjacent to a stream, or difficulties are encountered during the grouting operations. The operation then is as follows: A packer that can be expanded against the wall of the hole for high pressure and released after grout is set, is then placed; beginning at the first grout stop in the bottom of the drill hole, each crevice or fissure is grouted off to the point of no acceptance, or, to the desired pressure which was determined by use of the hydraulic sectional water-testing device.

Where an excessive amount of grout is accepted at low pressure, this is an indication of a direct leak outside of the shaft area. Blocking or building a barrier to keep grout within the shaft area then becomes necessary, since the object is to extend a water-tight plug, or, build a cofferdam around the outside of the shaft. Grouting operations are repeated by moving the packer the proper distance up the hole to the next packer stop or leak and continuing this until the entire hole is completely cemented. (Grout should be bypassed at each packer location holding an even pressure so that all openings are closed and grout will be retained where placed in the hole.)

The method above described is used for existing shafts into which water leaks during the summer and freezes in winter except that very low pressures must be maintained. It is not even safe to state the range of these pressures, due to the fact that some very weak shaft linings may be encountered. Detailed notes are kept of time, the ratio of cement used, the operating pressure, and possible back pressure from grout which has gone into openings higher in elevation. With these notes it can be determined when to change the mixture or to add certain quantities of swelling compound or chemicals to make a seal or when to bypass the mixture to close a leak.



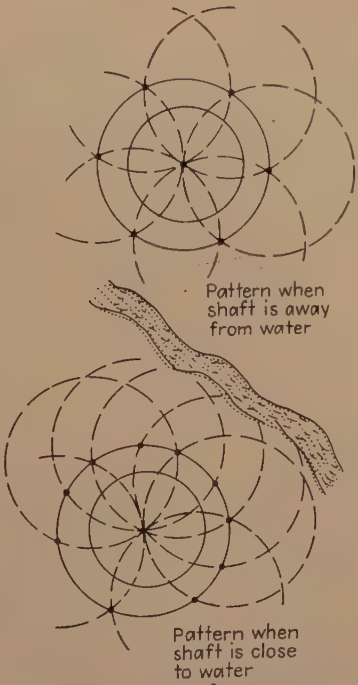

Pattern when shaft is away from water
Pattern when shaft is close to water

FIG 1*a*—PLAN. DIAMOND DRILL HOLES APPROXIMATELY 15° ANGLE IN DRAW SLATE AND INCLINED 15°.

FIG 1*b*—ELEVATION. DIAMOND DRILL HOLES APPROXIMATELY 15° ANGLE IN ROOF DETERMINED BY CONDITIONS.

When a large volume of water is encountered unexpectedly in driving a set of entries under a stream or broken section of ground, pumps and discharge facilities may not be available or of sufficient capacity, hence it may be good economy and less expensive, considering the cost of pumps, power and maintenance over a period of years, to consider grouting in taking care of water.

There have been various methods devised in laying out drill hole locations for such conditions. In this layout of holes, a broken water-bearing slate roof over the coal exists, ranging in thickness from 8 to 15 ft (Fig 2). On this plan, holes in the draw slate on both sides of the entry were drilled on a 15° angle and inclined 15° to the vertical, and one drilled upward in the center in line with the heading on the same inclined degree. The holes are then cored 40 to 50 ft ahead of the working face.

Grouting and testing of these angle holes are done by the same method as employed in grouting a shaft site.

Drill hole patterns or locations of holes should be thoroughly studied to dry up a wet entry, or, broken section when necessary to go under a stream, by drilling and grouting from the ground surface. The usual procedure is to drill directly in line with the heading. One line of holes may be sufficient. We have drilled as many as three lines, all going below the coal seam. This pattern shows a plan developed to give some idea of a spread and overlapping for different elevations in the drilled hole. Sealing the sidewalls and the roof does not always make the heading watertight—it may be necessary to grout with short holes in the floor of the mine. While no attempt has been made here to go into detail of this subject, its importance should not be overlooked in mining operations. This is only a preliminary outline of handling a grout mixture to build a watertight barrier surrounding a shaft location or a working area.

There are a large number of grouting jobs at coal and other mines which indicate a strong trend toward this permanent method of sealing water from mine openings and underground passages.

Mining engineers are more interested; shaft contractors are more interested; contractors interviewed are able to reduce their contract price when they can be protected by a water clause of only a few gpm and they can reduce their cost. We have grouted one shaft where it was perfectly dry.

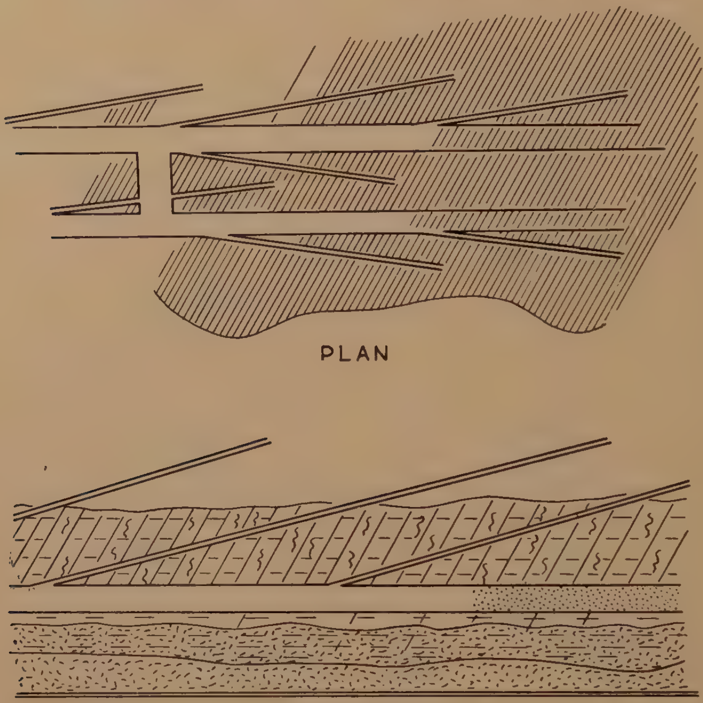


FIG 2—PROPOSED SHAFT DIAMOND DRILL HOLE PATTERN.

Grouting in Southeast Missouri District

By W. W. WEIGEL,* ASSOCIATE MEMBER AIME

IN "Mine Drainage, Southeast Missouri Lead District," *Trans.* AIME, (1943) 153, 74–81, the general water conditions in the lead mines of Southeast Missouri were described. Some comments developed from the experience of time and some further explanation and additions may now be made to that article.

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* St. Joseph Lead Company, Bonne Terre, Missouri.

The original article described how the mines, largely near the base of the 400-ft thick Bonne Terre dolomite underlain by a thick porous sandstone, get only a minor part of the inflow water through the sandstone by artesian circulation. The main intake points for the water handled at these mines are apparently where fracture and channel zones, which in some cases extend from the top to the bottom of the formation, cut under surface drainage lines. These allow large amounts of water to enter the mines either directly when the channel zones are cut by mine workings or else by keeping a slight head on the sand they force water up into the workings. In 1941, the average inflow of the district was 20,000 gpm, which with a 21,200 tons daily production on a six day week, gave a water-ore ratio of 6.5. Opening up of a large amount of virgin territory towards the outer edge of the district has increased this to a pumping load of approximately 25,000 gpm. Production is about the same, but on a five day week basis, so the water-ore ratio is now about 9.5 tons of water for each ton of ore.

Major pumping equipment has changed very little since 1941. Due to the necessity of developing low elevation ore, formerly ignored, the war years saw a considerable increase in the number of relay pumps.

No more areas have been isolated and bulkheaded from the mine as previously described. All previous work has held up well and has been entirely satisfactory. However, the peculiar circumstances of an isolated large wet area of mine workings with connections only through drifts do not often arise. The area mentioned as being shut off by three bulkheads has now been reopened. The plug type bulkheads are merely shot out and the drift is then clear. The lack of heavy reinforcement in the concrete can thus be a distinct advantage if it is desired to remove the bulkhead.

The "Mine Drainage" article described the mass grouting of large areas from the surface using fine flotation tailings spoken of as "slime." This has been done principally along the channel fracture zones and other main sources of water to the mine and has been highly effective. About 450,000 tons have been pumped, directly covering a total area of about 300 acres but affecting much more. During the latter part of the war, because of the stringent pipe situation and the shortage of labor, the program was shut down and has not been restarted. Several new water bearing areas which it would have been advisable to mass grout by "sliming" have opened but these have been too far from the mills (three to five miles). A new shaft recently sunk even farther (eight miles) from an active mill would have been an ideal case. Numerous large channels are in the area and merely sinking of the shaft caused several large sinkholes to form in the area. Several mud runs also occurred during the shaft sinking period and caused much trouble. No mining has been done from this shaft as yet.

During the war years, owing to pressure for production, open stope mining several hundred feet wide was carried across an intensely shattered area which had been one of the first to be mass grouted. Before sliming, even a drift through this had had to be abandoned after excessive water flows. The sliming with some follow-up underground cement grouting had so filled the openings in the area that water production is normally only about 500 gpm from this section.

This "follow-up" grouting work has been important. The sliming does not fill all the spaces and if water flows through these, part of the fine slime will be washed out, making the opening even larger. The grouting procedure in both slimed and unslimed areas is somewhat the same and is given in some detail below.

In drifting through areas where there is probability of water channels, it is customary to keep a pilot hole ahead. Long underground diamond drill holes require the shutting down of the drift and frequently have drilled through channels more or less mud-choked without showing evidence of them. Simpler and surer has been the practice of using a 22-ft hole drilled by the regular drift crew each round. By alternating from one side to the other and from top to bottom, these will give warning of the channel even if the angle with the drift is acute. On cutting a water channel with the pilot hole, the water is then shut off with a tapered wooden plug in the hole and preparations made for grouting the channel.

All loose rock is first cleaned off the face and then any small leaks through cracks are caulked up with wooden wedges and oakum. A hole (or several) with a diameter just under 2 in. is drilled into the face for a depth of about 3 ft and a grouting sleeve inserted. These are of 1½-in. pipe threaded on one end for a valve and with a slight taper for about 10 in. on the other end. A thin layer of friction tape is wound around the tapered end and the pipe driven into the hole with a hammer. If the face rock is broken or a very high pressure of water expected, the sleeve may be anchored with ties to anchor eye bolts to the side. A small hole is then drilled ahead of the sleeve with the steel passing through the valve and pipe until the water is cut. After withdrawal of the drill and closing of the valve, leakage of water in the face and around the sleeve may require more calking.

Actual grouting equipment consists of the pressure pumps, mixing barrels and connecting hose and pipe. No. 6 Cameron pumps are used, with an 8-in. air cylinder and with water cylinder bushed down to 3½ in. With 90 lb air pressure this gives approximately 400 lb pressure for grouting. As the maximum hydrostatic head is less than 600 ft, there is ample capacity. Mixing barrels used are round tanks, 36 in. diam by 30 in. high, with a semi-cone bottom. An Ingersoll-Rand No. 25 sump pump is placed in each barrel as an agitator (the discharge of the sump pump is directly back into the barrel). Two barrels are used. Each barrel mix consists of 5 to 10 sacks of cement for about 70 gal of water. While one barrel is pumped the other one has its batch prepared. The entire outfit is mounted on flat cars and should be pushed as close as possible to the face to be grouted; that is, less than 35 ft. Connections are made from the grout pump to the sleeves with high pressure 2-in. pipe and hose with union ends. A blow-off valve is placed in the line as close to the face as possible.

Actual order of grouting varies somewhat. If the channel in the face is thought to have a large amount of thin mud, clear water may be pumped at first to get room for the cement grout. If the face is badly shattered and broken with many leaks, a small amount of very thick grout may be pumped slowly and then allowed to set. Occasionally fine mule feed and a large amount of aquagel may be required as an addition. A new grouting hole is then prepared. It may be necessary to repeat this and gradually build up the rock so that blowouts of grout will not occur. If the face is solid and the channel apparently open, regular pumping is started with a mix of about 5 sacks of cement to 70 gal of water, gradually thickening the successive mixtures up to about 10 sacks per 70 gal until the pump finally stalls.

Common cement only is used. Aquagel is used as an addition of from one to two pounds per sack of cement. This allows much thicker mixtures to be pumped without strain on the pump and the grout sets better to the shape of the channel. The use of any fine sand has been found to be detrimental as there is usually separation in the channel. Temperature of the water is 58°F. Pumping once started is continuous. Four men on a shift have pumped as high as 300 sacks. As one hole shuts off, another one is started if ready, until the face will not take any more. After the channel is completely grouted, the cement is allowed to set for about a week. The drift is then advanced carefully through the channel until firm rock is met on the other side or more water is hit in a second channel.

The amount of grout pumped into any one channel bears little relation to the initial pressure. Three or four sacks or as high as 1500 sacks may be needed. In one drift that was crossing a channel zone, fifteen channels of various sizes were cut in a distance of about 1000 ft. All had close to 135 lb initial pressure and flows up to 1200 gpm. Grouting averaged about 500 sacks per channel (some up to 1000) and did good sealing. It should be noticed particularly that this area had not been mass grouted by sliming from the surface. The fifteen grouting operations, totaling some 8000 sacks of cement, in a slimed area would be reduced to possibly nine or ten grouting operations and would not use more than several hundred sacks of cement. In slimed areas, initial pressure of channels is quite low, rarely as much as 50 lb, and normally cement grouting will not require more than 30 to 40 sacks at the most. Many of the channels as cut are already completely blocked. The slime grouting and cement grouting are thus seen to be good mutual complements. As the slime has already thoroughly settled and compacted, there is no mixing with the cement grout. The cement fills the remaining small openings and shuts off the last trace of water. The slime grouting cuts the amount of cement grouting to a mere fraction of that required without it. In addition, as it covers a wide area comparatively cheaply, it is not restricted to making a small ring around a narrow opening such as a drift, but is suitable for preparing shattered and fractured areas for large mine openings such as high breast stopes.

DISCUSSION

I. B. Crosby*—The writer recently has had an opportunity to compare European with American grouting methods and he was impressed with the much greater use of chemical grouting in Europe. A few of the typical uses will be mentioned. It is used for tightening finely fractured rock, for producing impervious grout curtains in sand, and for grouting about tunnels in sand. It was used to produce a grout curtain in fractured limestone under the Castillon Dam, now being completed, on the Verdon River, Southern France. It was used also in producing a grout curtain under the existing earth dam at Lac Noir, in the Vosges Mountains, France, where the grout was injected in sand and gravel of a glacial moraine. Experiments and tests have just been completed for a grout curtain more than 300 ft deep in fine alluvium at the proposed Serre-Poncon Dam on the Durance River in Southern France. Mixtures of sodium silicate and sodium aluminate with varying amounts of cement and bentonite were used in each case. An example of tunnel grouting is the work on one of the Paris subways to prevent leakage where it passed through fine sand. A mixture of sodium silicate and sodium aluminate was used and successful results were obtained. Chemical grouting has been used successfully at many other places in Europe for various purposes and also in Algiers, Africa.

The successful use of chemical grouting in other countries raises the question as to whether more consideration should not be given to its use in similar problems in this country.

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Barrier Pillars in the Anthracite Region of Pennsylvania*

By S. H. Ash† and W. L. Eaton‡ Members AIME

(New York Meeting, March 1947)

Introduction

A barrier pillar in the anthracite region of Pennsylvania is a portion of the bed that is left unmined along the property line or lines of adjoining mining properties, or between mines, or between parts of mines.

Barrier pillars perform important functions in anthracite-mining operations. Their principal function is to act as dams to prevent water that accumulates in a mine from suddenly breaking into an adjacent mine or into adjacent mine workings and causing loss of life, property, or both.

More than 83 billion gal of water is impounded in 115 underground water pools in anthracite mines.[1,2] These pools are confined by a system of barrier pillars which separate adjacent mines.

The principal factor that threatens to cut short the life of the anthracite industry, to curtail production, and to affect the economic structure of the people and business dependent on anthracite for their livelihood is the inundation of anthracite mines.[1,2,4,13,29]

To obtain a clear picture of the anthracite mine-water problem, it is necessary to understand the part played by the barrier pillars that separate adjoining mines.

Purpose of Report

The purpose of this report is to show the methods of determining the size of barrier pillars in the anthracite region, to present the hazards that exist because of the lack of sound engineering being applied to the size of many existent barrier pillars, and to suggest a method whereby barrier pillars can perform the function for which they are intended.

The Barrier-pillar Problem

Article III, section 10, of the Anthracite Mining Laws of Pennsylvania reads thus:

> It shall be obligatory on the owners of adjoining coal properties to leave, or cause to be left, a pillar of coal in each seam or vein of coal worked by them, along the line of adjoining property, of such width that, taken in connection with the pillar to be left by the adjoining property owner, will be a sufficient barrier for the safety of the employees of either mine in case the other should be abandoned and allowed to fill with water; such width of pillar to be determined by the engineers of the adjoining property owners together with the inspector of the district in which the mine is situated, and the surveys of the face of the workings along such pillar shall be made in duplicate and must practically agree. A copy of such duplicate surveys, certified to, must be filed with the owners of the adjoining properties and with the inspector of the district in which the mine or property is situated.

Mining practices with regard to barrier pillars have varied greatly since mining began. In many anthracite mines the barrier pillars have been weakened or punctured. As a consequence, the condition of barrier pillars in many abandoned mines and parts of some active mines is unknown.

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* Published by permission of the Director, Bureau of Mines, U.S. Department of the Interior. Manuscript received at the office of the Institute Jan. 17, 1947. Issued as TP 2289 in Coal Technology, November 1947.

† Chief, Safety Division, U.S. Bureau of Mines, Washington, D. C.

‡ Mining Engineer, Safety Division, Bureau of Mines, Wilkes-Barre, Pennsylvania.

[1] References are at the end of the paper.

It is beyond the scope of this report to describe the means of protection being taken by anthracite-mining companies to prevent the failure of a barrier pillar if the hydrostatic pressure against it exceeds a definite figure.[2]

At present the Federal Bureau of Mines is conducting a study on each barrier pillar to ascertain, if possible, whether or not it is large enough to withstand safely the hydrostatic pressure against it if the mine fills with water.

Not much is known[3] about barrier-pillar seepage. A study on water pools underground in the anthracite region is too recent, and information is too meager, to have developed many data on this question. Other than knowledge that such seepage adds to the average water inflow on a property, there appears to be nothing that can be done about this seepage as long as water is impounded underground.

METHODS OF DETERMINING SIZE OF BARRIER PILLARS

The width of a barrier pillar between the mine workings of adjoining properties in the anthracite region is determined by the engineers of the adjoining property owners and the State mine inspector in whose district the properties are situated.

An arbitrary rule for determining the width of barrier pillars, adopted by a number of mining companies and the State mine inspectors of the Pennsylvania anthracite districts, is known as the "Mine Inspectors' Formula for Barrier Pillars." The rule is as follows: "Multiply the thickness of the coal bed, in feet, by 1 pct of the depth below the drainage level, and add to this five times the thickness of the coal bed."[9]

This rule can be written in formula, as follows:

$$W = (T \times 0.01 \times H) + 5T,$$

when W = width of barrier pillar, in feet; H = depth below drainage level, or hydrostatic head, in feet; and T = thickness of the bed, in feet.

The drainage level of a mine is that altitude above which water will not rise. It is the overflow point and may be the surface altitude of a shaft or slope or the point at which an outside tunnel or drift intersects the coal measures. It may also be an opening in a barrier pillar that allows water to drain from one mine into another.

Thus, for a bed 6 ft thick and 400 ft below the drainage level, the barrier pillar should be, according to this rule,

$$(6 \times 0.01 \times 400) + (5 \times 6) = 54 \text{ ft wide}$$

Using this rule, or formula, computations were made to determine the widths of barrier pillars that are required for all beds of anthracite that range in thickness from 3 to 30 ft, to withstand a hydrostatic head varying in height from 0 to 800 ft.[2] This information is shown in Fig 1.

By consulting Fig 1, the barrier-pillar width, when computed by the Mine Inspectors' Formula, can readily be determined for any combination of bed thickness and hydrostatic head.

To determine the widths of barrier pillars, other conditions besides the thickness of the bed and the vertical distance below the drainage level are considered by engineers and State mine inspectors.

The nature of the bed in which the barrier pillar is to be established must be considered carefully. If it is crushed by folding or is friable, it may not be satisfactory. Although it may not fail through collapse, its seepage factor may be too high; this results in excessive pumping costs to a company operating a mine adjacent to one that is filled with water.

The geologic structure of the coal measures where the barrier pillar is to be established must be carefully studied. It is evident that a barrier pillar established in an area in which a fault is present may prove useless. Water might readily flow along the fault and pass from one side of the barrier pillar to the other.

The dip of the bed also plays an important part in the determination of the width of a barrier pillar, whether the proposed pillar be perpendicular to the strike of the coal measures or parallel to it.

Where the beds are flat, or nearly so, and not friable or faulted, the width of the barrier pillar is generally determined by applying the Mine Inspectors' Formula; however, where the beds dip steeply, the results obtained from the formula are arbitrarily increased by the engineers and the State mine inspectors, according to their judgment and experience.

The method generally employed to mine steeply dipping beds is to drive tunnels through the rock strata and anthracite beds, on a slight upgrade and perpendicular to the strike so as to intersect all the beds in the area. A gangway, or entry, is then driven in each bed, and the rooms (chambers, or breasts) are driven from the gangway. The area that consists of the gangways and rooms from them and is mined from a given tunnel is called a level. There may be one or a number of levels, depending on the size of the mine and the depth of the syncline (synclinal axis) of the coal measures.

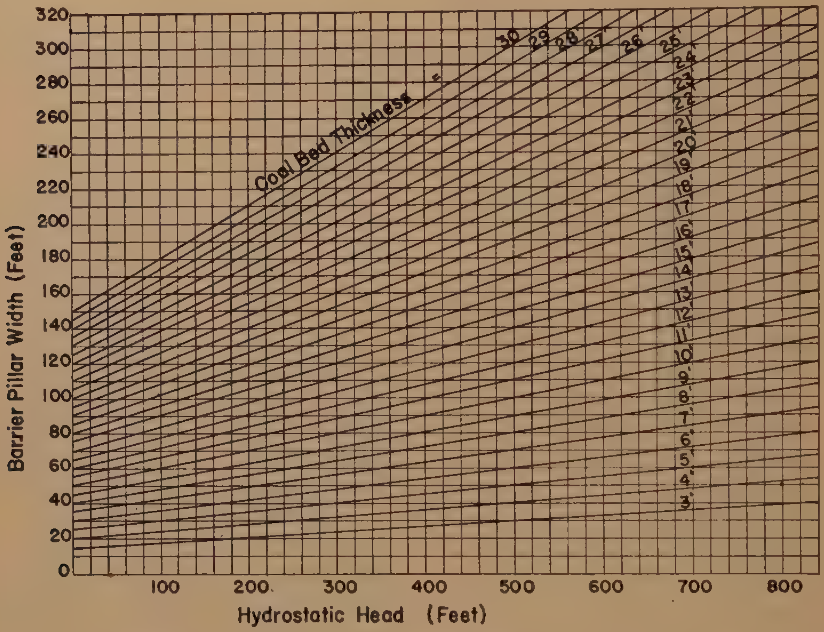


FIG 1—BARRIER PILLAR WIDTHS COMPUTED FROM MINE INSPECTORS' FORMULA.

The width, or size, of the barrier pillar applying to the barrier pillars in the respective beds that are mined on a given level, where the beds dip steeply, is based on the thickness of the thickest bed intersected by the tunnel and is calculated by the Mine Inspectors' Formula. In a given level, this width applies to all the beds, irrespective of their thicknesses; therefore, it is the width to be maintained between the given tunnel and the bottom of the tunnel of the next-above level.

As the barrier-pillar width applying to the beds mined on each level is calculated by the Mine Inspectors' Formula, and, as the thickness of the thickest bed is approximately the same in each level, the cross section of the strata constituting a barrier between the mine workings adjacent to the barrier assumes the form of a pyramid by becoming wider with each successive level of lower altitude. A barrier having a pyramidal cross section is much more stable than one having a cross section such as is obtained by the application of the Mine Inspectors' Formula to several beds where the coal measures dip gently or are flat. (Fig 6.)

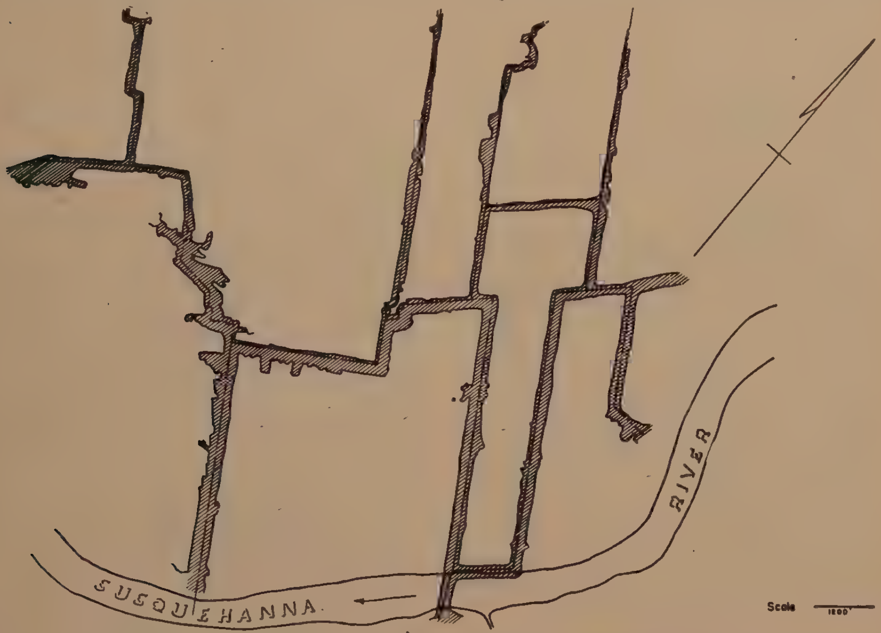


FIG 2—PLAN MAP OF TYPICAL BARRIER PILLARS NORTHERN ANTHRACITE FIELD—PENNSYLVANIA.

To prevent damage by mining subsidence to railroads, highways, buildings, and surface areas, where pillars are extracted in the beds under them, a large anthracite-mining company has established a procedure to govern the width of protective pillars. The protective pillars are designed so that the sides of a barrier consisting of the rock strata and anthracite beds under the surface area to be protected, are theoretical lines that are 45° to the horizontal from the surface to the top of the rock formation and 70° to the horizontal downward from the top of the rock formation to the lowest bed in which pillars are extracted.

On the map that shows the mine workings of an individual bed, some anthracite-mining companies have plotted points that designate places that, in their opinion, are weak. A notation at each of these points gives the width of the barrier pillar as it should be, the actual width, and the altitude.

Where a barrier pillar acts as a dam between an active mine and an inactive mine that is gradually filling with water, such a barrier pillar often is considered by the mining company to be unsafe as a protection against a hydrostatic head that is more than a definite figure. In such an active mine, holes are driven through the barrier pillar at strategic points. Valves are attached to the bore-holes to control the hydrostatic pressure against the barrier pillar. At critical points, the barrier pillars have been strengthened by large blocks of masonry, reinforced concrete, hydraulic backfilling, or by a combination of these means of protection.

Some Typical Barrier Pillars

Although a plan map of the mine workings containing a barrier pillar is necessary to show the hazards created by hydrostatic pressure against the face of the barrier pillar, it is also necessary to show a cross section of the strata constituting the coal measures that overlie the barrier pillar.

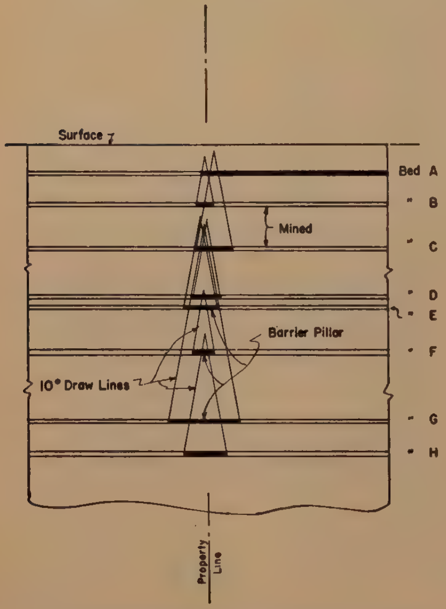


Fig 3—Cross section of an actual barrier showing 10° draw lines from each bed.

Fig 2 is a plan map of some typical barrier pillars between adjacent mines working the same bed in a part of the Wilkes-Barre area.

Fig 3 and 4 show cross sections of the strata containing some typical barrier pillars separating adjacent mines in the northern anthracite field.

A barrier that is to act as a dam must be designed to prevent subsidence of the critical strata or coal measures that overlie the barrier pillars of all the beds contained in the proposed barrier between the mines affected.

In this report, the authors use the term *draw line* to mean a line drawn from the respective face, or edge, of the barrier pillar in the bed in which the anthracite is extracted to the edge of a surface area that may be affected by mining subsidence. The term *draw line* is synonomous with the term *limiting line* used by Briggs.[7]

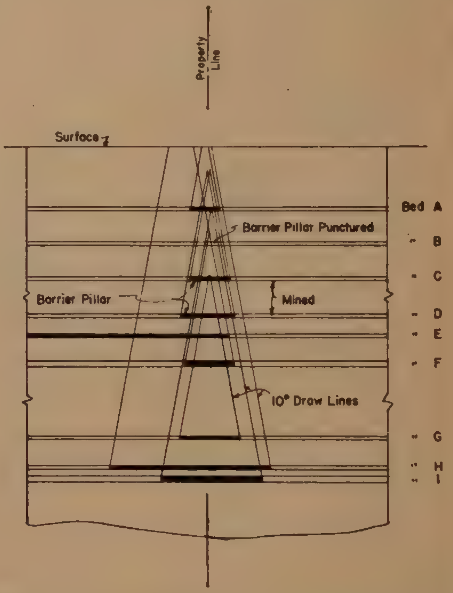


Fig 4—Cross section of an actual barrier showing 10° draw lines from each bed.

They also use the term *the angle of draw* to mean the angle of intersection of the draw line and a vertical line drawn from the respective edge of the barrier pillar to the surface. The draw line, with respect to the vertical line, is considered in this report as being over the barrier pillar.

To prevent a squeeze and maintain the stability of a barrier that is designed to act as a dam between adjacent mines, the strata on both sides of the barrier must bend, break, or both at points that are outside of or coincide with the faces of the barrier. These faces must be outside of or coincide with the above-defined draw lines.

Experience has proved that the horizontal projection of the width of a barrier pillar should be more than the width of any horizontal section of the overlying strata that the barrier pillar is intended to support. Consequently, the angle of draw should be assumed to be no less than 10°[7,15,19,20,21,23].

Draw lines having angles of draw of 10° are shown in Fig 3 and 4 to illustrate the possible effects of mining subsidence on the existent barrier pillars, where the beds are partly or completely extracted on both sides of the barrier pillars.

If the width of the barrier pillar in each bed, where several beds are mined, is not sufficient, the draw lines of some of the barrier pillars may intersect the barrier pillar of one or several of the overlying beds, or intersect each other before they reach the surface or the altitude of the drainage level of the mine. Wherever the draw lines of a given barrier pillar intersect at a point below the surface or below the drainage level, subsidence of the surface or drainage facility will occur.

The cross sections of the two actual barriers between adjacent mines, shown in Fig 3 and 4, definitely indicate that damage to the barrier pillars can be anticipated. In almost every instance the barrier pillar in each bed either is intersected by the draw line or lines of subsidence of the lower beds or is over the point or points of intersection of draw lines.

In some of the beds shown, mining has been conducted on only one side of the barrier pillar. In one bed it is known that the barrier pillar has been punctured.

Discussion of Barrier Pillars

Where mining operations are conducted in more than one bed, the barrier pillars separating adjacent mines in which the same beds are mined are not always placed in a vertical plane, or columnized, but are offset.

There are reasons why barrier pillars may be offset from each other. The ownership of the various beds may be different, and property lines may overlap; the beds may have been mined at different times; or an overlying bed may have been considered nonminable at the time an underlying bed was extracted, and the extraction of the underlying bed caused damage to the overlying bed or beds, which were mined afterward.

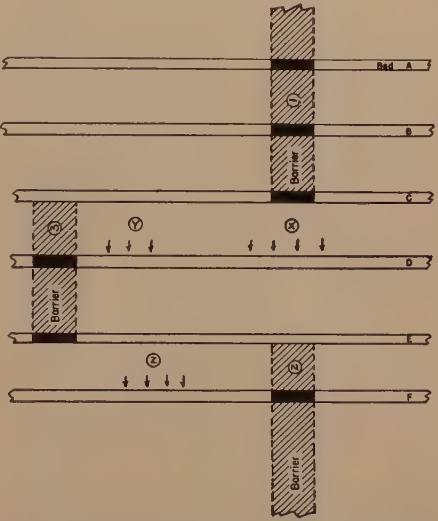


Fig 5—Cross section through offset barrier.

The offsetting of barrier pillars is open to criticism, because a number of potentially dangerous conditions can result, especially if the barrier pillars are expected to act as dams. As a consequence, protection is not provided that will prevent water from flowing into mine workings on either side of the barrier pillars, if the mine workings are interconnected by broken strata or otherwise.

Fig 5 illustrates some of the dangerous conditions that may arise if barrier pillars are offset.

The weight of the strata, which includes the barrier pillars in beds *A*, *B*, and *C*, and which is shown as Barrier 1, is supported by the strata underlying bed *C*.

When bed *D* is extracted to the boundary of the strata shown as Barrier 3, the strata above bed *D* may break and cause a cave, as at *X*. This can result in the failure of Barrier 1, by subsidence. If water were impounded in mine workings on either or both sides of Barrier 1, especially in mine workings in bed *C*, the water would flow into mine workings in bed *D* and could result in damage to the mine and possible loss of life to persons working therein. If the strata at *Y* or *Z* break, water would flow from one side to the other of the barriers intended for the protection of mine workings in beds *D*, *E*, and *F*, and the barrier pillars, therefore, would be useless. The mine workings in bed *E* provide an overflow point over Barrier 2 for water impounded in the mine workings of bed *F*.

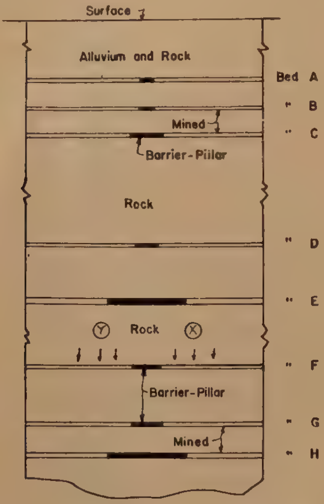


FIG 6—COLUMNAR SECTION THROUGH THEORETICAL BARRIER-GENTLY DIPPING BEDS.

A barrier consisting of the strata that contain barrier pillars designed in accordance with the Mine Inspectors' Formula may be unstable, where the coal measures contain several beds that dip gently or are flat, because the beds have different thicknesses, and it so happens that the thickest bed is not the lowest bed in the coal measures.

Fig 6 illustrates the form of the columnar section obtained where the barrier pillars are designed strictly in accordance with the Mine Inspectors' Formula. It shows that the strata composing such a barrier between mine workings would be of doubtful stability where the beds are completely extracted adjacent to the barrier pillars.

Where bed *F* is mined, especially if completely extracted on one or both sides of the barrier pillar, subsidence of the overlying strata is inevitable. The extent of the subsidence cannot be predicted. Caving may extend upward to Bed *E*, as shown in *X* and *Y*, Fig 6, and result in severe damage to the barrier pillar in bed *E*. See also Fig 3 and 4.

Subsidence of any part of the barrier pillar in bed *E* would undoubtedly create a weakness in this barrier pillar. If the mine on either side of the barrier pillar were abandoned at a later date, and the abandoned mine workings in bed *E* were allowed to fill with water, a risk of inundation would exist in the active mine on the opposite side of the barrier pillar. The weakened barrier pillar in bed *E* might collapse and either cause sudden inundation of the active mine, with probable loss of life and possibly the loss of the mine, or allow large quantities of water to seep through the barrier pillar and into the active mine.

If the barrier pillar is perpendicular to the strike of the bed, or perpendicular to the synclinal axis, the gangways or passageways driven on the strike are perpendicular to the pillar, or nearly so, and the rooms (chambers, or breasts) are parallel to the pillar. When the rooms adjoining the barrier pillar are finished, and particularly if the room pillars are extracted, the barrier pillar may run, break off, or slough. This causes a pocket which might extend through the barrier pillar.

Fig 7 shows possible damage to a barrier pillar where the rooms are driven parallel and adjacent to the barrier pillar.

If the barrier pillar is parallel to the strike, the synclinal axis, or the anticlinal axis of the coal measures, the gangways or passageways driven on the strike are parallel to the pillar, and the mine workings driven off the gangway are perpendicular to the pillar. It is evident that here, too, when the rooms are finished, the barrier pillar may run, break off, or slough. This results in the barrier pillar being weakened, if not actually punctured.

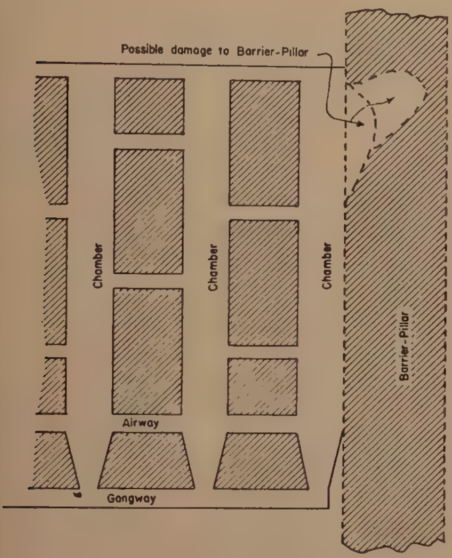


Fig 7—Barrier pillar perpendicular to the strike of the coal measures—steeply pitching beds.

Fig 8 shows possible damage to a barrier pillar where the rooms (chambers, or breasts) are perpendicular, or nearly so, to the barrier pillar.

When establishing barrier pillars parallel to the strike of the coal measures, mining companies endeavor to place them exactly in the axis of the syncline, or basin. This lessens the probability of damage to the barrier pillar. It is not always possible to do this, because of different ownership of the beds or because the exact position of the synclinal axis may not be known until the mine workings are well-advanced.

From the foregoing discussion on the determination of the size and position of a barrier pillar and on the conditions of existent barrier pillars, it is apparent that uncertainty exists in the minds of those concerned, not only as to the method by which the size of a barrier pillar is determined but also as to the stability of a barrier pillar that is to serve as a dam.

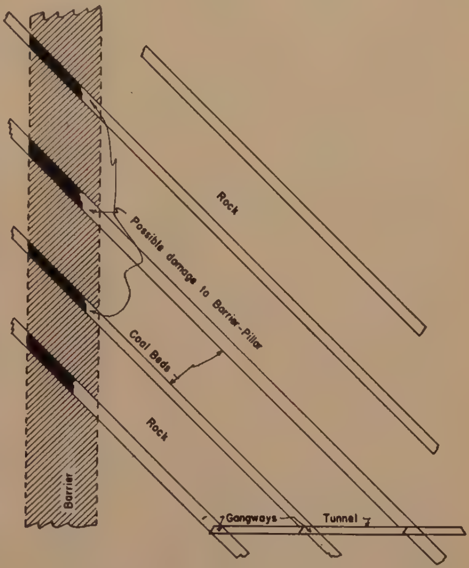


Fig 8—Barrier pillars parallel to the strike of the coal measures—steeply pitching beds.

Such a state of uncertainty regarding the correct size of a barrier pillar results in a pillar that either is excessively large or is unconsciously made too small. Where a barrier pillar must serve as a dam, the failure of it may result in loss of life, the mine, or both, and it would be far better had the barrier pillar not existed and thereby given a false sense of security.

A Proposed Standard Method to Determine the Size of a Barrier Pillar that Acts as a Dam

The question of barrier pillars is intimately associated with the questions of ground movement (mining subsidence, squeezes, and rock bursts) and load strength (roof support).

To deal adequately with the technical details of the above-mentioned questions is beyond the scope of this report. However, unless mining operations are conducted so as to prevent a squeeze or rock burst (bump) that would affect a barrier pillar having the correct size to serve as a dam, there is no assurance that the stability of the barrier pillar can be maintained.

The subject of ground movement is still an interesting and controversial one, particularly the questions of mining subsidence, rock bursts, and the behavior of strata as their depth below the surface of the earth increases.[22]

Briggs[7] has discussed the theories of mining subsidence and gives a summary on conclusions regarding the mechanics. Herbert and Rutledge[15,30] and Young and Stoek[31] have given information on mining subsidence in American practice that is pertinent in designing barrier pillars. Landsberg[17] prepared a bibliography on mining subsidence that, although incomplete, comprises more than 800 titles of articles on the subject. Ash and Westfield,[3] Griffith and Conner,[14] Montz and Norris,[24] and Moore and Powell[25] have discussed mining subsidence in the anthracite region.

The question of rock bursts has been recently discussed by Bucky and others,[8] by Irving,[16] and by Phillips, Messervey, and Casey.[26]

Ashmead,[5] Darton,[10] Dierks,[11] Enzian,[12] Griffith and Conner,[14] Montz and Norris,[24] and Rice and Enzian[28] have discussed ground movement and roof control in reports that deal solely with anthracite problems.

The reserves of virgin anthracite lie at considerable depth in the anthracite region. Unless something is done to solve the mine-water problem in this region, it is only a matter of time until the mine workings in many deep anthracite mines will be underwater workings. The condition of barrier pillars in both abandoned mines and active mines becomes increasingly important. The barrier pillars on which the safety of active mines depends can be severely damaged by squeezes, subsidence, or rock bursts.

The largest percentage of the anthracite now being produced in underground anthracite mines comes from pillars. Because underground mining is and has been conducted by room-and-pillar methods, breaks in the roof have attained major importance. Furthermore, as pillars are extracted in mines that may not even be close to some particular mine, large areas in that mine become affected by squeezes that override the barrier pillars. Because of the system of mining in anthracite mines, breaks in the roof extend over barrier pillars, and a squeeze can render a barrier pillar useless in a very short time.

Where pillars have been left near a barrier pillar, and the mine workings have not been backfilled, water or water-laden strata that are controlled by the barrier pillar become a serious threat to the safety of the mine and persons employed therein, in the event of a squeeze and the failure of the barrier pillar.[6] The same hazard often exists when pillars are extracted near a barrier pillar some time after the rooms are driven and the worked-out area is not backfilled as the pillars are extracted.[6] This is because of roof breaks and damage to the barrier pillar resulting from room-and-pillar mining.

The authors believe that the stability of barrier pillars that are correct in size, when designed, can be maintained by the complete extraction of the bed being mined, preferably by a longwall method, and tightly filling the worked-out area by hydraulic or pneumatic backfilling with sand or similarly sized materials.[3,27]

Behavior of a Barrier That Acts as a Dam

It has been previously stated that the principal function of a barrier pillar in anthracite-mining practice is to act as a dam and control water in underground pools. Such being the case, it would seem logical that the ideal solution to the problem of barrier-pillar design would be to consider that a barrier pillar is part of a dam, or barrier, which is composed of the overlying strata, or coal measures. Such a dam, or barrier, would react in the same manner as a masonry dam. The term *masonry dam* will be used in this discussion.

There are two ways, or cases, in which a masonry dam may resist the thrust of the water: (1) By the inertia of its masonry and (2) as an arch.

A barrier that is composed of the above-given coal measures is definitely not an arch, where it acts as a dam; consequently Case 2 need not be considered.

Case 1. The horizontal thrust of the water is held in equilibrium by the resistance of the masonry; consequently, the barrier must neither slide forward nor overturn. Such a dam is called a *gravity dam.*

Failure of a gravity dam can occur in three ways:

1. By sliding along a horizontal joint, or plane, or shearing along any section.
2. By overturning.
3. By the crushing of the material composing the dam.

Because the materials comprising a barrier that is designed to act as a dam are the natural strata, the resistance of the barrier to crushing is enormous.

When a gravity dam is designed, the dimensions of the structure are so made that the resultant of forces passes through the middle third of its base; this gives the structure a factor of safety of approximately 3. If the resultant of forces passes outside the toe of the free face of the dam, the dam will overturn when it is fully loaded.

The faces of a barrier that acts as a dam will never be entirely free, the barrier will receive lateral support even though mining has been completed on both sides of it, and the subsequent draw lines of subsidence are established. It is logical, therefore, to give the barrier, or dam, such dimensions that the resultant of forces (considering the cross section of the barrier and the thrust of the water alone and not the other materials adjoining the face of the barrier) intersects the base of the barrier, or dam, just on the inside of its outer edge. The barrier will then be stable and have a factor of safety of 1 against overturning. Additional factors of safety are given by the resistance offered by the materials adjoining the sides of the barrier.

The cross-section of a barrier that acts as a dam has the form of a trapezoid, of which the nonparallel sides make equal angles with the vertical, because the direction of the water thrust can not be predetermined and depends entirely on which mine is abandoned first and allowed to fill with water. Therefore, the barrier, to act as a dam, must be designed to resist water thrust from either side. A barrier having a trapezoidal cross section will satisfy these conditions.

Determination of Correct Size of Barrier That Acts as a Dam

It was found by trial calculations that a barrier, or dam, having a trapezoidal cross section with a top width of 50 ft, arbitrarily chosen, and a base width 0.45 of the height will satisfy the condition that the resultant of forces intersects the base of the barrier just on the inside of its toe and consequently will be stable.

The height of the trial barrier was assumed to be 1130 ft, which is known to be the depth of the coal measures in an anthracite property. The sides of this barrier are at an angle of 11° 27′ with the vertical, or approximately 1 horizontal in 5 vertical. The trial barrier, or dam, therefore satisfies the condition against overturning and against sliding. The factor of safety against overturning is 1 and against sliding is 1.4.

The coefficient of friction of the material composing the barrier was taken as 1.

Comparison of Stability of Barrier That Acts as a Dam with Stability of Barrier Designed by Mine Inspectors' Formula

The cross section and stability of the strata composing the above-mentioned barrier that acts as a dam, can be compared with the cross section and stability of the strata composing the coal measures that contain the barrier pillars having widths that are calculated by applying the Mine Inspectors' Formula, as follows:

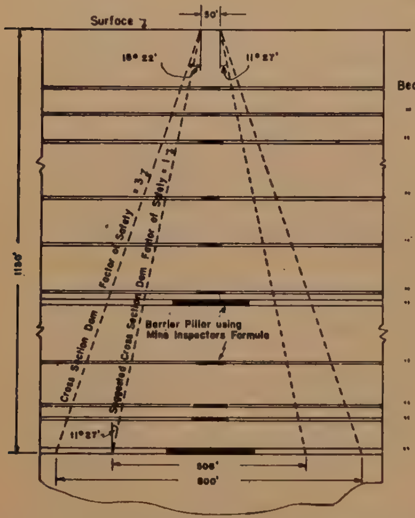


Fig 9—Cross section of barrier, or dam, superimposed on columnar section containing barrier pillars designed according to Mine Inspectors' Formula.

The depths, below the surface, of the anthracite beds in the place chosen for the trial dam are known. By considering the thicknesses of the beds to be the average thicknesses of the given beds in this particular area, the widths, or sizes, of the respective barrier pillars are calculated by substituting the real values of the depths and thicknesses for the symbols in the Mine Inspectors' Formula. (See p. 63.)

The calculations are not given; however, for comparison, the cross section of the above-mentioned barrier, or dam, can be superimposed on a figure showing the columnar section of strata containing the barrier pillars having widths that are calculated by applying the Mine Inspectors' Formula. This is shown in Fig 9.

The barrier, or dam, shown in Fig 9, has a trapezoidal cross section. Such a barrier will not be damaged by mining subsidence, if the angles that its sides make with the vertical lines drawn from the toes of the base of the barrier, or dam, are equal to or more than the angles of draw, as defined on page 66.

The angle of draw varies widely, as reports in the literature show.[7,15,19,23,31] The authors believe that the angle of draw should not be assumed to be less than 10°. This is for safety, and because of the wide range (0° to 90°) in the magnitude of the angle of dip of beds in the anthracite region as well as the tendency of pillars to "run," where the beds dip steeply.

As the cross section of the barrier, or dam, shown in Fig 9, has sides that make an angle of 11° 27′ with the vertical, or approximately 1 horizontal to 5 vertical, such a barrier would not be damaged by subsidence due to the complete extraction of the bed in the areas adjoining the barrier pillars, provided that the areas in which the beds are extracted are large enough, and mining is conducted in such a manner that a squeeze or bump can not occur and the strata overlying the mined-out areas can subside.

By assuming that an angle of draw of 10° applies to the barrier pillars shown in Fig 9, we can construct Fig 10 by plotting the respective draw lines on Fig 9.

Fig 10 has been constructed to show the instability of barrier pillars that have been designed in strict accordance with the Mine Inspectors' Formula and are to act as dams, as compared with the stability of the barrier pillars where they are component parts of a barrier that is designed to act as a dam.

Fig 10 definitely indicates that damage to the barrier pillars, which have widths designed in accordance with the Mine Inspectors' Formula, can be anticipated. The barrier pillars, except those in the three lower beds, can be seriously affected, if not destroyed, by subsidence. Without exception, they are too small to act as dams, if either mine were filled with water. On the other hand, if the barrier pillars were component parts of a barrier designed to act as a dam, the strata on both sides of this barrier would bend or break at draw lines that would be outside of the faces of the barrier, and neither the rock strata nor the barrier pillars that compose the barrier would be affected by subsidence or hydrostatic pressure.

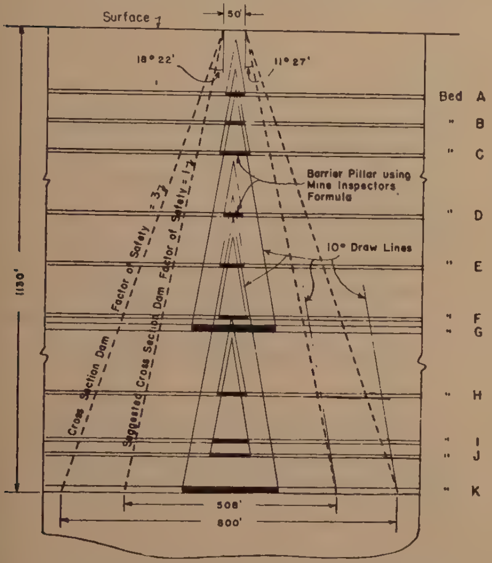


FIG 10—CROSS SECTION OF BARRIER, OR DAM, SUPERIMPOSED ON COLUMNAR SECTION CONTAINING BARRIER PILLARS DESIGNED IN STRICT ACCORDANCE WITH THE MINE INSPECTORS' FORMULA AND ALSO SHOWING 10° DRAW LINES FROM EACH BED.

Fig 3 and 4 are cross sections of the strata containing actual barrier pillars separating adjoining properties, where the widths of the barrier pillars were determined by the engineers of the companies and state mine inspectors concerned. To what extent the Mine Inspectors' Formula was used to calculate the widths, or sizes, of the barrier pillars is not known, except that it was in accordance with anthracite-mining practice.

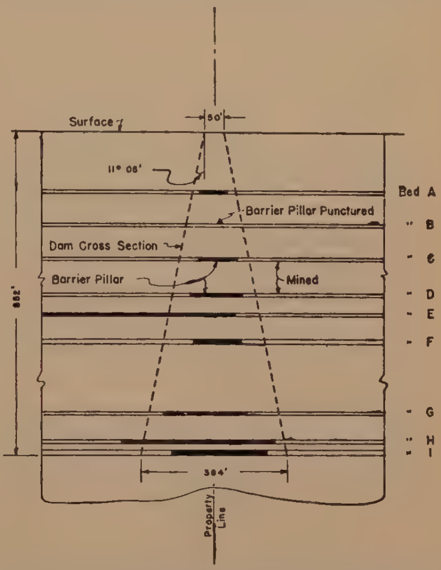


FIG 11—COLUMNAR SECTION CONTAINING ACTUAL BARRIER PILLARS ON WHICH IS SUPERIMPOSED A CROSS SECTION OF BARRIER DESIGNED TO ACT AS A DAM.

Because a stable barrier consisting of rock strata and barrier pillars in the beds that are shown in Fig 3 is presumed to act as a dam, such a barrier must be designed to withstand a hydrostatic head of 786 ft, which is the vertical distance between the surface over the barrier and the floor of the lowest bed (*H*, Fig 3). The arithmetical calculations are not given; however, the sides of the barrier are found to make an angle of 10° 57′ with the vertical. The length of the base of the barrier, or width of the barrier pillar in bed *H*, is 354 ft. The barrier, or dam, has a coefficient of friction of 0.70 and a factor of safety of 1.43 against sliding, and a factor safety of 1 against overturning.

The differences between the widths of the existent barrier pillars and the widths of the barrier pillars that are component parts of a barrier designed to act as a dam are shown graphically in Fig 11, which is constructed by superimposing the cross section of the barrier on the columnar section that contains the barrier pillars shown in Fig 3.

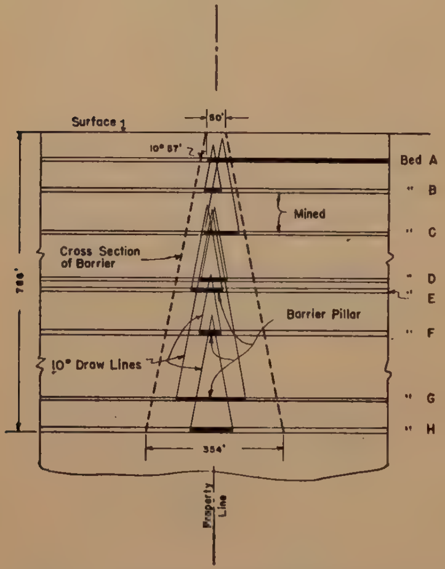


FIG 12—CROSS SECTION SHOWING ACTUAL BARRIER PILLARS, 10° DRAW LINES FROM THE BARRIER PILLARS IN EACH BED, AND BARRIER DESIGNED TO ACT AS A DAM. SCALE: ONE INCH = 200 FT.

We have assumed that an angle of draw of 10° applies to the existent barrier pillars shown in Fig 3 and 11. Fig 12 is constructed either by plotting the respective draw lines on Fig 11 or by superimposing Fig 3 on Fig 11.

Fig 12 shows the instability of the actual barrier pillars, if they are to act as dams in the event that either mine adjoining the barrier pillars is filled with water, as compared with the stability of the barrier pillars where they are component parts of a barrier designed to act as a dam.

Fig 12 indicates definitely that damage to the existent barrier pillars can be anticipated. The draw lines of subsidence indicate that, with the exception of the barrier pillar in the lowest bed *H*, the barrier pillars are likely to be affected by subsidence. With the exception of the barrier pillar in bed *A*, they are too small to act as dams if either mine is filled with water. However, the critical part of the barrier pillar in bed *A* is in strata that are likely to subside and consequently cannot be depended upon to prevent water that may be impounded in the mine workings in bed *A* from entering into the mine workings in bed *B*.

It is also seen in Fig 12 that, if the barrier pillars were component parts of a barrier designed to act as a dam, the draw lines of subsidence due to the mined-out areas adjoining the sides of the barrier would be outside of the faces of the barrier, and neither the rock strata nor the barrier pillars composing the barrier would be affected by either subsidence or hydrostatic pressure.

As a stable barrier consisting of rock strata and barrier pillars in the beds that are shown in Fig 4 is presumed to act as a dam, such a barrier must be designed to withstand a hydrostatic head of 852 ft, which is the vertical distance between the surface over the barrier and the floor of the lowest bed *I*, Fig 4. The arithmetical calculations are not given; however, the sides of the barrier are found to make an angle of 11° 05′ with the vertical. The length of the base of the barrier, or width of the barrier pillar in bed *I*, is 384 ft. The barrier, or dam, has a coefficient of friction of 0.71 and a factor of safety of 1.41 against sliding, and a factor of safety of 1 against overturning.

The differences between the widths of the existent barrier pillars and the widths of the barrier pillars that are component parts of a barrier designed to act as a dam are shown graphically in Fig 13, which is constructed by superimposing the cross section of the barrier on the columnar section that contains the barrier pillars shown in Fig 4.

We have assumed that an angle of draw of 10° applies to the existent barrier pillars shown in Fig 4 and 13. Fig 14 is constructed either by plotting the respective draw lines on Fig 13 or by superimposing Fig 4 on Fig 13.

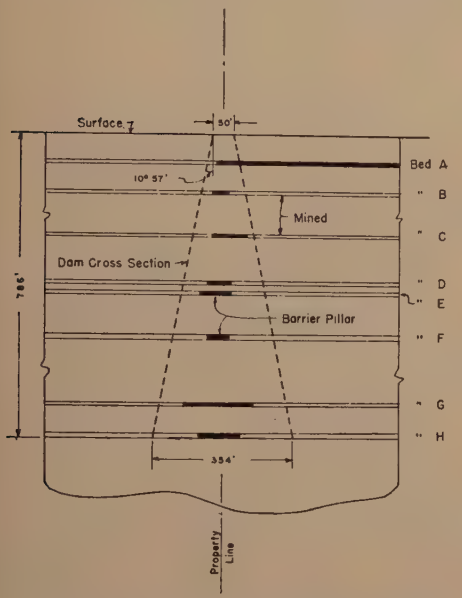


FIG 13—COLUMNAR SECTION CONTAINING ACTUAL BARRIER PILLARS ON WHICH IS SUPERIMPOSED A CROSS SECTION OF BARRIER DESIGNED TO ACT AS A DAM.

Fig 14 definitely indicates that damage to the existent barrier pillars can be anticipated. The draw lines of subsidence indicate that, with the exception of the barrier pillar in the lowest bed *I*, the existent barrier pillars are likely to be affected by subsidence. None of them can be depended upon to act as a dam in the event that either mine adjoining the barrier pillars is filled with water. This is because the barrier pillars are too small, they may be damaged by subsidence, and their stability is uncertain. Moreover, the barrier pillar in bed *A* is useless because no barrier pillar exists in bed *B*. Although the barrier pillar in bed *H* is wide enough, the portion of bed *I* that is under the barrier pillar in bed *H* has been extracted.

It can also be deduced from Fig 14 that, if the barrier pillars were component parts of a barrier designed to act as a dam, the draw lines of subsidence due to the mined-out areas adjoining the sides of the barrier would be outside of the faces of the barrier, and neither the rock strata nor the barrier pillars composing the barrier would be affected by the subsidence or hydrostatic pressure.

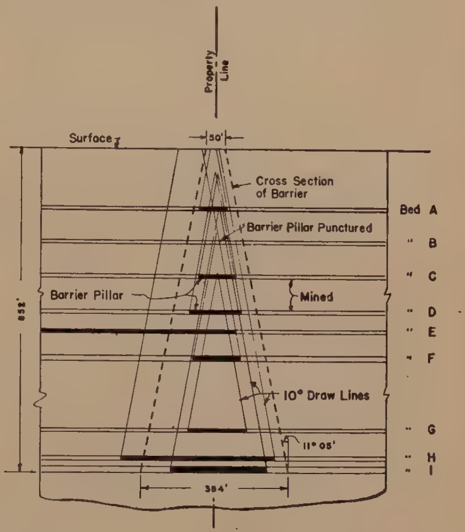
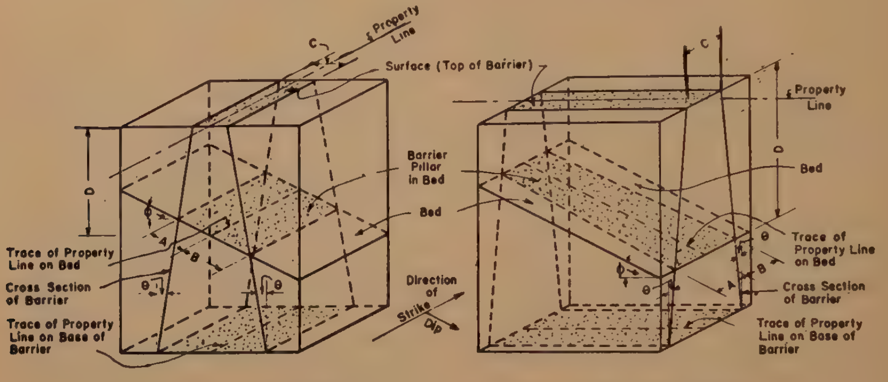


FIG 14—CROSS SECTION SHOWING ACTUAL BARRIER PILLARS, 10° DRAW LINES FROM THE BARRIER PILLARS IN EACH BED, AND BARRIER DESIGNED TO ACT AS A DAM.

A Formula for Determining Width of a Barrier Pillar

Mining practice supports the opinion that any rule, or formula, for the size of a barrier pillar is to be regarded merely as a guide. No rule can possibly serve in all instances; however, a barrier pillar is of vital importance where it must serve as a dam to protect a mine and the persons employed therein.

The prevention of failure of a barrier pillar that is acting as a dam requires a remedy that leaves no question of doubt. Such a remedy requires the use of a simple formula by which the width, or size, of a barrier pillar can be determined without attempting to connect the thickness of the bed and the width of the barrier pillar, by means of a formula.



a- Property Line and Barrier Pillar are Parallel to Strike of Bed

b- Property Line and Barrier Pillar are Perpendicular to Strike of Bed

FIG 15—PROCEDURE FOR DERIVING FORMULA, BY WHICH THE WIDTH, OR SIZE, OF BARRIER PILLAR IS CALCULATED.

a. Property line and barrier pillar are parallel to strike of bed. b. Property line and barrier pillar are perpendicular to strike of bed.

To protect a barrier pillar, whether the bed is steeply inclined or flat and the size of the barrier pillar is ample to serve as a dam, the mined-out areas adjoining the barrier pillar must be tightly filled, preferably with sand or similarly sized materials that are placed hydraulically or pneumatically.

The dimensions of barriers to act as dams have been calculated, where the hydrostatic heads to be resisted are 786, 852, 1130, and 2000 ft, respectively.

The cross section of a barrier to resist a hydrostatic head of 2000 ft, which is the approximate maximum depth below the surface reached by the lowest workable bed in the anthracite region, has the following dimensions: Depth, 2000 ft; top, 50 ft; and base, 900 ft. The sides of the barrier have a slope of 12° to the vertical. The factor of safety against sliding is 1.35 and that against overturning is 1.

Fig 15 illustrates the procedure for deriving the formula for determining the width, or size, of a barrier pillar in a bed, regardless of the thickness of the bed and its inclination, or dip.

If in Fig 15 we express all measurements of distance in feet, and let

A = the width on the plane of the bed, of the portion of the barrier pillar on the rise side of the property line;

B = the width, on the plane of the bed, of the portion of the barrier pillar on the dip side of the property line;

$L = A + B$ = the width, or size, of the barrier pillar;

C = the horizontal width of the barrier, or dam, as measured on the surface or at the drainage level; this is considered to be 50 ft, which the authors believe should be the minimum width of the top of a barrier that acts as a gravity dam;

D = the vertical distance, or depth, between the property line on the surface and the trace of the property line on the floor (bottom, or footwall) of the bed;

d_{rise} = the horizontal distance from the property line to the floor of the bed, where the bed outcrops at some point on the rise side of the property line on the top of the barrier, or dam;

d_{dip} = the horizontal distance from the property line to the floor of the bed, where the bed outcrops at some point on the dip side of the property line on the top of the barrier;

θ = the angle that the nonparallel sides of the cross section of the barrier make with the vertical, where the barrier has a top that is 50 ft wide, has a depth of no more than 2000 ft, and has a factor of safety against sliding of not less than 1.35 and against overturning of not less than 1. This angle is taken as 12° for a barrier, so that it can serve as a gravity dam;

φ = the dip of the bed.

We can derive the following equations that are applicable where the bed does not outcrop within the limits of the top of the barrier, or dam, and the property line and barrier pillar are parallel to the strike of the bed (see *a*, Fig 15):

$$A = \frac{\left(\frac{C}{2} + D\tan\theta\right)[\sin(90^\circ - \theta)]}{\sin(90^\circ + \theta - \varphi)}, \qquad (1)$$

where C = 50 ft, and θ = 12°, or

$$A = \frac{24.45 + 0.208D}{\sin(102^\circ - \varphi)}; \qquad (2)$$

$$B = \frac{\left(\frac{C}{2} + D\tan\theta\right)[\sin(90^\circ + \theta)]}{\sin(90^\circ - \theta - \varphi)}, \qquad (3)$$

where C = 50 ft, and θ = 12°, or

$$B = \frac{24.45 + 0.208D}{\sin(78^\circ - \varphi)}; \qquad (4)$$

$$L = A + B, \qquad (5)$$

where C = 50 ft, and θ = 12°, or

$$L = (24.45 + 0.208D)\left(\frac{1}{\sin(102^\circ - \varphi)} + \frac{1}{\sin(78^\circ - \varphi)}\right). \qquad (6)$$

If the inclination of the bed is zero (a flat bed), or $\varphi = 0$,

$$A = B = \frac{C}{2} + D\tan\theta, \qquad (7)$$

where C = 50 ft, and θ = 12°, or

$$A = B = 25 + 0.213D, \qquad (8)$$

and

$$L = A + B = 50 + 0.426D. \qquad (9)$$

Where the bed outcrops on the rise side of the property line on the top of the barrier, or dam, we find that

$$A = \frac{d_{\mathrm{rise}}}{\cos\varphi}, \qquad (10)$$

and B is found by means of Eq 4.

Where the bed outcrops on the dip side of the property line on the top of the barrier, or dam, we find that

$$A = 0, \qquad (11)$$

and

$$B = \frac{\left(\frac{C}{2} - d_{\mathrm{dip}}\right)[\sin(90^\circ + \theta)]}{\sin(90^\circ - \theta - \varphi)}, \qquad (12)$$

where C = 50 ft, and θ = 12°, or

$$B = \frac{24.45 - 0.978\, d_{\mathrm{dip}}}{\sin(78^\circ - \varphi)}. \qquad (13)$$

Where the property line and barrier pillar are perpendicular to the strike of the bed (see *b*, Fig 15), and, regardless of the thickness of the bed, its inclination, and whether or not it outcrops, the portions, A and B, of the base of a given cross section of the barrier are of equal magnitude with respect to the vertical plane of the property line.

The depth D, or vertical distance between the surface or drainage level and a definite point that is formed by the intersection of a given vertical cross section of the barrier and the trace of the property line on the plane of the floor of the bed, is equal to the vertical distance from the surface to the floor of the bed, which is considered as being flat, regardless of its inclination. In other words, φ is considered as being zero at the above-mentioned point in the bed.

The mine-property owner is interested to know how much of the given bed must be left on his side of the property line. The widths *A* and *B* of the portions of the barrier pillar can be calculated by means of whichever equation (1 to 13 inclusive) applies to the problem; however, the real values of the widths *A* and *B* can also be determined by means of Fig 16 and 17.

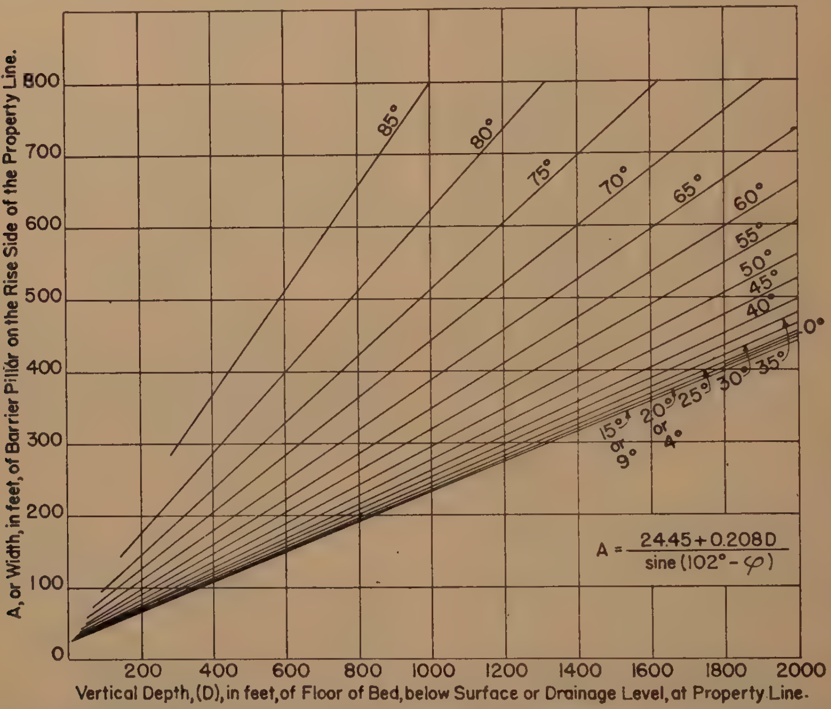


FIG 16—WIDTHS OF PORTIONS OF BARRIER PILLAR, ON THE RISE SIDE OF THE PROPERTY LINE, WHERE THE BED DOES NOT OUTCROP ON THE TOP OF THE BARRIER, OR DAM, AND THE BARRIER PILLAR AND PROPERTY LINE ARE PARALLEL TO THE STRIKE.

Because of the above-given reasons, the real value of *A* or *B*, where the property line and barrier pillar are perpendicular to the strike of the bed, can be found by substituting the real value of *D* in Eq 8, or

$$A = B = 25 + 0.213D. \qquad (8)$$

Where the beds dip steeply, the widths *A* and *B* applying to the barrier pillar are governed by the position of the barrier between the properties concerned. To recover a justifiable portion of a bed, the position of the barrier, with respect to the property lines, may be changed by the property owners concerned.

CONCLUSIONS

From the foregoing discussion on the existent barrier pillars in the anthracite region of Pennsylvania, and particularly with regard to the lack of strict observance of the Mine Inspectors' Formula for determining the widths of barrier pillars, it is apparent that uncertainty exists in the minds of all concerned, not only as to the adaptability of any rule for the size of a barrier pillar, but also as to the stability of many existent barrier pillars which are presumed to act as dams.

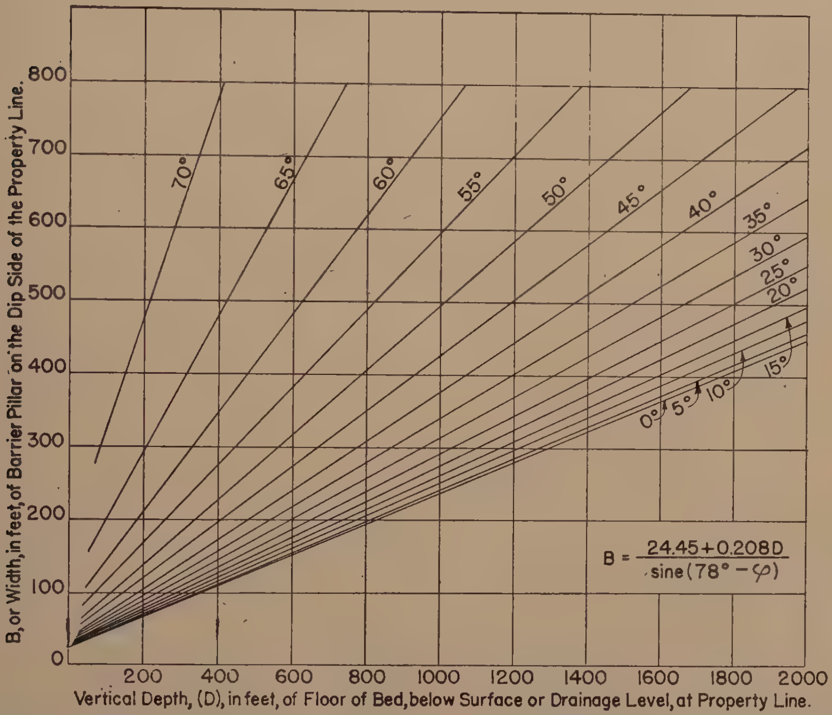


FIG 17—WIDTHS OF PORTIONS OF BARRIER PILLAR, ON THE DIP SIDE OF THE PROPERTY LINE, WHERE THE BED DOES NOT OUTCROP ON THE TOP OF THE BARRIER, OR DAM, AND THE BARRIER PILLAR AND PROPERTY LINE ARE PARALLEL TO THE STRIKE.

By considering that the strata containing barrier pillars is a dam, and by designing the barrier as a gravity dam, the authors believe that the barrier pillars, extensive in size though they are in deep mine workings, are of such importance as to justify their size.

Often barrier pillars are punctured by passageways in which masonry dams are constructed to resist hydrostatic pressure. If the barrier pillar itself is unstable because of ground movement or lack of load strength, a masonry dam in any unstable part of the barrier pillar can fail even before the barrier pillar collapses.

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DISCUSSION

CADWALLADER EVANS, JR.*—The paper clearly indicates that the authors are apprehensive about the water problems in the anthracite region, which have been under study by the engineers of the coal companies and of the State for many years, and by the U. S. Bureau of Mines for a part of that time.

These present-day problems would be very much simplier if the need and value of barrier pillars had been realized 75 years ago, when mining at depth was getting under way in this region. Unfortunately anthracite-bearing lands were not generally controlled in large blocks by one company or individual, and because of the irregular shape of the holdings the amount of coal which would have had to be left for barrier pillars of the size which this paper advocates, was too great to be considered, taking into account the economics of the industry at the time.

The State Mine Inspectors developed a formula for barrier pillars which the authors of this paper consider inadequate. There is a possibility that they are right, but it must be remembered that the formula was not intended to be, and in general has not been interpreted and applied literally. It is a guide to determine minimums, and in every case the known or expected conditions of the surrounding strata should be taken into account in determining actual sizes. In many cases that come to our knowledge, these factors have been taken into account when the barrier pillar was formally agreed upon by the State Mine Inspectors and the engineers of the companies involved, prior to its being legalized in a formal "Barrier Pillar Agreement." The great difficulty in the past has been that barrier pillars were not established soon enough. As a consequence, excavations were made too close to property lines before there was any attempt to establish a barrier pillar and, obviously, all that could be done was to make the best of what existed.

The discussion proposed by the authors is interesting and we may accept their assumptions as being correct, but it must be remembered that before they can be applied, there must be careful evaluation of the character of the roof, the character of the coal bed itself, and of the underlying strata. Consideration must be given to the nature of the overburden some distance above the roof, because this frequently unknown but important factor has caused trouble in the past.

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The paper will be of value in guiding determinations regarding the size of barrier pillars in the southern part of the anthracite field, but seems to be academic with respect to the larger portion of the anthracite field, since these barriers are already largely determined and their extent limited by mining already done.

If barriers of the size recommended in this paper are to be established, it will be necessary for the companies to pool their holdings in order to lay out colliery operations ample enough to justify barrier pillars of the size which the authors indicate to be necessary. To make economically possible the use of such barrier pillars, the land lines separating the properties would have to be readjusted so as to run across the valleys instead of following the measures as they frequently do now.

Oil and Gas Wells Drilled through Workable Coal Seams

By Arch J. Alexander*

(New York Meeting, February 1948)

Coal is produced, in commercial quantities, in thirty-five of the fifty-five counties of West Virginia. Oil and gas are produced, commercially, in forty-two counties. So, you may readily see that coal, oil and gas are closely related as to production areas. Many of the West Virginia coal mines are studded with oil and gas wells, as drilling has been done in thirty-two of the coal bearing counties.

When the drilling of oil and gas wells was placed under the regulatory control of the Department of Mines on June 1, 1929, it was estimated, by the State Geological Survey, that 64,000 wells had been drilled in West Virginia and that 34,000 wells were then being operated as oil and gas producers. No records of many of these remaining 30,000 abandoned wells exist and some of them remain a standing menace to safety as our coal extraction continues.

Of a total of 13,100 new wells drilled since June 1929, approximately 5200 have been drilled through coal measures. Approximately 7000 wells, which were drilled through coal measures, have been abandoned in the same period of time. Many of these wells were drilled prior to the enactment of the oil and gas laws.

Prior to the enactment of the Oil and Gas Law, which established the Oil and Gas Division of the Department of Mines in 1929, there had been several mine explosions attributable to cutting into uncharted wells and the improper plugging of other wells. The most serious explosion was that of Mines No. 47 and No. 49 of the Consolidation Coal Co. in Marion Co., when one of the so-called plugged wells leaked or seeped through some 80 ft of strata to the coal bed elevation, filling the mine with gas, the gas being ignited, killing a good many men and wrecking two coal mines.

Legislation, to combat well hazards to coal mines, was slow in coming. In 1928, R. M. Lambie, then Chief of the Dept. of Mines, got sufficient support from coal, oil and gas interests to get the legislature to pass our present oil and gas laws. These statutes not only protected the coal industry but went a long way toward conserving the oil and gas resources of our state.

Before drilling for oil or gas on any tract of land, known to be underlaid with one or more workable seams of coal, the well operator is required to submit a map of the area showing the location of the proposed well, and to send a copy to the Dept. of Mines and a copy to the coal operator. If no objections to the location of the proposed well are made by the coal operator within ten days, a drilling permit and a number for the well are then issued and work may proceed. If, however, the location of the proposed well is objected to by the coal operator, on the grounds that the location of the well will interfere with present mine operations, projected workings, or otherwise cause a dangerous condition, the operator may file a written objection with the Dept. of Mines.

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If the well operator and the coal operator, between themselves, cannot agree on a new location suitable to both parties, then either party can demand a hearing to be held in the office of the Chief of the Dept. of Mines and he decides the issue. However, either party, if dissatisfied with the decision of the Chief of the Dept. of Mines, can appeal to the Circuit Judge of the district in which the proposed well is to be located.

The method of procedure in drilling through workable coal beds is, quoting from the law: "A well penetrating one or more workable coal beds shall be drilled to such depth, and of such size, as will permit the placing of casing and packers in the hole at such points and in such manner as will exclude all oil, gas, or gas pressure from the coal bed, except such as may be found in the coal bed itself. Each string of casing run in the hole shall be provided with a steel casing shoe or collar firmly fixed on the bottom of the string of casing. Each string of casing run through a workable bed of coal shall be seated, at least thirty feet below such coal bed, in twenty feet of cement, mud, clay or such other nonporous material as will make an effective seal, and after any such string of casing has been so seated, drilling may proceed forthwith to any required depth." It has been the practice, in the northern part of the state, to cement this largest casing through the coal and when abandoned, is left in the well with a vent to the outside.

Protective devices when gas is found beneath or between workable coal beds, again quoting from the law: "In the event that gas is found beneath a workable coal bed before the hole has been reduced from the size it had at the coal bed, a packer shall be placed below the coal bed, and above the gas horizon, and the gas by this means diverted to the inside of the adjacent string of casing through perforations made in such casing, and through it passed to the surface without contact with the coal bed. Should gas be found between two workable beds of coal, in a hole of the same diameter from bed to bed, two packers shall be placed, with perforations in the casing between them, permitting the gas to pass to the surface inside the adjacent casing. In either of the cases here specified, the strings of casing shall extend from their seats to the top of the well."

There are many cases when wells are drilled through areas in a coal mine that has been mined out. The law provides as follows: "When a well is drilled through the horizon of a coal bed from which the coal has been removed, the hole shall be drilled at least thirty feet below the coal bed, of a size sufficient to permit the placing of a liner which shall start not less than twenty feet above it. Within this liner, which may be welded to the casing to be used, shall be centrally placed the largest sized casing to be used in the well, and the space between the liner and casing shall be filled with cement as they are lowered into the hole. Cement shall be placed in the bottom of the hole to a depth of twenty feet to form a sealed seat for both liner and casing. Following the setting of the liner, drilling shall proceed in the manner provided above. Should it be found necessary to drill through the horizon of two or more workable coal beds from which the coal has been removed, such liner shall be started not less than twenty feet below the lowest such horizon penetrated and shall extend to a point no less than twenty feet above the highest such horizon."

When a well is to be abandoned, the well operator shall notify the coal operator and again quoting from the law: "Prior to the abandonment of any well, the well operator shall notify, by registered mail, the coal operator or operators to whom notices are required to be given by section two of this article, and the Department of Mines, of its intention to plug and abandon any such well (using such form of notice as the Department may provide), giving the number of the well and its location and fixing the time at which the work of plugging and filling will be commenced, which time shall be not less than five days after the day on which such notice so mailed is received or in due course should be received by the Department of Mines, in order that a representative or representatives of the coal operator and of the department, or of both, may be present at the plugging and filling of the well. Whether such representatives appear or do not appear, the well operator may proceed at the time fixed to plug and fill the well in the manner hereinafter described. When such plugging and filling have been completed, an affidavit, in triplicate, shall be made (on form to be furnished by the department) by two experienced men who participated in the work, in which affidavit shall be set forth the time and manner in which the well was plugged and filled. One copy of this affidavit shall be retained by the well operator, another (or true copies of same) shall be mailed to the coal operator or operators, and the third to the Department of Mines. A marker of concrete or iron shall be erected over the location, with the name of the company and number of the well stamped or cast upon it as a means of identification to the coal operator or any other interested persons.

"Upon the abandonment or cessation of the operation of any well drilled for natural gas or petroleum, the well operator, at the time of such abandonment, or cessation, shall fill and plug the well in the following manner:

"Where the well has penetrated one or more workable coal beds, it shall be filled and securely plugged in the manner aforesaid, to a point forty feet below the lowest workable coal bed. If, in the judgment of the well operator, the coal operator and the Department of Mines, a permanent outlet to the surface is required, such outlet shall be provided in the following manner: A plug of cement, or other suitable material, shall be placed in the well at a suitable point, not less than thirty feet below the lowest workable coal bed. In this plug and passing through the center of it shall be securely fastened an open pipe not less than two inches in diameter, which shall extend to the surface. At or above the surface of the pipe shall be provided a device which will permit the free passage of gas, and prevent obstruction of the same. Following the setting of the cement plug and outlet pipe as aforesaid, the hole shall be filled, with cement to a point twenty feet above the lowest workable coal bed. From this point the hole shall be filled with mud, clay or other nonporous material to a point thirty feet beneath the next overlying workable coal bed, if such there be, and the next succeeding fifty feet of the hole filled with cement, and similarly, in case there are more overlying workable coal beds. If, in the judgment of the well operator, the coal operator, and the Department of Mines, no outlet to the surface is considered necessary, the plugging, filling and cementing shall be last above described."

Up to now only the side of the picture that protects the coal operator has been presented. The oil and gas operator has some rights in this matter, also. When a coal operation extends its working within 500 ft of any well already drilled or being drilled, a map shall be filed with the Dept. of Mines showing its mine workings, projected workings and proposed pillar plan, and at the same time, a copy mailed to the well operator. The size of the pillar, for the protection of the well, depends on the depth of the overburden. It has been the practice to leave a maximum pillar of 200 ft square. This pillar must be left permanently.

Following the filing of such plans and map, the coal operator may proceed with the operations to within 200 ft of the well. If there are no objections by the well operator, a permit is issued by the Dept. of Mines to the coal operator to extend mine workings, according to plans on file.

The coal operator shall, at least every six months, bring such plans or map so filed up to date.

The well operator has the right to file objections to the coal operator driving closer than 200 ft and hearings are held in the office of the Chief of the Dept. of Mines in the same manner as locating a new well.

The law further provides that the Dept. of Mines shall have supervision over all drilling and mining operations and shall make such rules and regulations for both the well operator and the coal operator bearing on this specific subject. The Dept. of Mines has received very fine cooperation from both coal and oil and gas operators. There were many uncharted wells drilled prior to 1929, and both coal and oil and gas operators have gone to a great deal of expense in trying to locate these uncharted and abandoned wells on their properties. There have been instances where cutting machines have cut into the casing of these uncharted wells but, fortunately, no gas was encountered as only the outer casing was penetrated and, in old unplugged wells, the water pressure had shut off the gases. There have also been instances where wells have been improperly plugged and have caused some difficulty, such as permitting top hole water to drain into the mine. At the present time, due to a scarcity of oil and gas well casing, many wells are being plugged and there are quite a few contractors who are doing this work for the oil and gas producers. It might be well to know that, at this time, some of these people, when they encounter difficulties, such as parted casings, cave-ins, and the like, will neglect to carry out the details of the work order and they bear careful watching. The "Work Order," which is prepared from the well record, outlines the procedure to be followed when plugging a well.

The oil and gas division of the Dept. of Mines is set up as follows: Its head is a practical oil and gas man with many years experience in the business, with the title of Administrative Assistant. He has one inspector-at-large and four inspectors in the field. These men are all oil and gas men with many years experience and we feel that the office is well administered. It is the job of the five inspectors to see that the oil and gas wells are properly plugged in accordance with the work order issued by the oil and gas division.

The drillers of all wells shall keep an accurate and complete log or record of all wells drilled, showing all formations drilled through, the casing used and where set; where water, fresh and salt, was encountered and the amount; where oil and gas were encountered, amount and volume and any other information that may be of value to the future development of the state's natural resources. The well operator is required to file with the Dept. of Mines this "data sheet" or "well record" when drilling is completed or when wells are to be plugged.

Permanent records of all the wells drilled and plugged since 1929, together with pillar plans for mining around wells, are now on file with the oil and gas division, along with maps of their location and other data, providing for the mine operators, oil and gas operators, geologists and other interested persons, a comprehensive and complete record of some 23,000 wells.

Maintenance of Coal Mining Equipment. Report of Maintenance Committee—Coal Division AIME

By A. Lee Barrett,* Chairman, Member AIME

(New York Meeting, February 1948)

The Maintenance Committee of the Coal Division of the American Institute of Mining and Metallurgical Engineers is presenting as a part of this report the second of a series of maintenance cost comparisons. The committee's purpose is to assemble maintenance costs on all equipment connected with mechanical operations at the face of coal mining and to arrange these costs by districts so that comparisons can be made between districts and within districts.

The committee, this year, sent 35 questionnaires to representative coal companies who either reported last year or had advised specifically of the inability to report to last year's questionnaire primarily because of limited time. The response to the questionnaire this year was not as good as last year, only 20 of the 35 selected companies returning a completed questionnaire.

This year's data have been compiled and combined with last year's data to present a comparison of the 1945 and 1946 costs of loading machines, shortwall cutting machines, universal cutting machines, service locomotives, shuttle cars, and mine cars. The same tabulation by districts is used as was used for 1945. As was indicated by last year's report there is considerable variation in the maintenance cost of mining machinery in the various coal mining districts and due to this high variability it was not felt practical to attempt to arrive at weighted averages. Consequently, all average figures used in this report are straight averages.

An analysis was made of companies reporting both this year and last year in an attempt to determine the trend in maintenance costs for different classes of equipment. Ten representative companies are used in this computation. Table 1 shows the average cost for 1945 and 1946.

Table 1—*Maintenance Cost Trend*
1945–1946

Type of Equipment	1945 Cost	1946 Cost	Per Cent of Increase 1946 over 1945
Mobile Loaders........	0.065	0.075	+15.4
Shortwall Cutters......	0.037	0.047	+27
Universal Cutters......	0.058	0.055	− 5
Post Mounted Drills...	0.011	0.014	+27
Service Locomotives...	0.035	0.043	+22.9
Shuttle Cars..........	0.048	0.067	+39.5
Mine Cars............	0.013	0.016	+23

It is felt that the negative trend in the maintenance cost of universal cutting machines probably represents the effect of the purchase of a relatively large number of new machines in 1946 with little maintenance cost appearing during the first year of operation. The grand total figure indicates an upward trend amounting to plus 18.7 pct. However, this figure is viewed with some suspicion because of the effect of the universal cutters decreasing maintenance cost and it is concluded that direct application of percentage increase should be made to types of equipment.

In Table 2 covering maintenance costs of various types of mining equipment for 1945 and 1946 the results obtained in Table 1 indicating trends were applied to those

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1945 figures where companies did not report in 1946, so as to arrive at 1946 maintenance cost for such companies. The committee felt that while such corrections could not be depended on for individual operations, in overall averages they would be quite safe.

TABLE 2—*Maintenance Cost of Mining Equipment* 1945–1946

	Loaders		Shortwall Cutters		Universal Cutters		Service Loco.		Shuttle Cars		Mine Cars	
	1945	1946	1945	1946	1945	1946	1945	1946	1945	1946	1945	1946
Central Pa	0.083	0.095	0.038	0.048			0.041	0.050	0.082	0.115	0.019	0.023
Western Pa	0.078	0.092	0.030	0.038	0.064	0.062	0.040	0.049	0.053	0.070	0.015	0.018
Virginia	0.085	0.073	0.051	0.063	0.030	0.027	0.038	0.061	0.099	0.168	0.015	0.015
West Virginia	0.073	0.084	0.037	0.046	0.060	0.052	0.059	0.071			0.023	0.028
Eastern Ky	0.055	0.056	0.039	0.050	0.045	0.052	0.050	0.062	0.155	0.158	0.018	0.019
Western Ky	0.057	0.072	0.051	0.072	0.044	0.042	0.017	0.021	0.045	0.059	0.004	0.005
Illinois	0.069	0.080	0.028	0.035	0.047	0.045	0.033	0.040	0.076	0.107	0.016	0.019
Indiana	0.063	0.072	0.045	0.057	0.056	0.053	0.027	0.033	0.060	0.084	0.005	0.006
Ohio	0.091	0.110	0.045	0.054	0.090	0.085	0.037	0.047	0.040	0.049	0.011	0.013
Alabama	0.089	0.103	0.055	0.070	0.047	0.045	0.051	0.063	0.069	0.097	0.032	0.039
Wyoming	0.039	0.042	0.029	0.033			0.014	0.010	0.010	0.015	0.014	0.017
Montana	0.102	0.118	0.050	0.063			0.066	0.081			0.038	0.047

Table 2 was compiled from 33 companies reporting for 1945 and 1946 and is felt to be representative with respect to the locations indicated. An attempt was made this year to establish a definite basis of costs as a result of comments at the time of the presentation of the reports last year, namely, to base all costs on material handled. Because of the smaller number of responses received this year it was not considered feasible to set the cost up on this basis. We might point out, however, that in the case of those companies reporting this year, costs were on a clean coal basis in the ratio of 14 to 1. It seems reasonable to assume, therefore, that companies reporting in 1945 would hold to about the same ratio and the table in general could be considered as based on clean coal.

The amount of spare equipment maintained was much more uniform between companies this year than was the case last year.

It will be noted that the number of active loading machines per spare last year and this year are about the same.

Data this year indicate that time between overhauls has been lessened by approximately 30 pct. This was indicated by a general trend and probably represents greater experience with mechanized mining.

An analysis of the relative cost of labor and material in maintenance cost reveals that 45 pct of the total maintenance cost in 1946 was labor and 55 pct was material.

TABLE 3—*Spare Equipment for Various Classes of Equipment*

TYPE OF EQUIPMENT	NO. OF ACTIVE MCHS. PER SPARE
Loaders	5
Shortwall cutters	6
Universal cutters	7
Locomotives	17
Shuttle Cars	14

TABLE 4—*Average Time in Shifts between Overhauls*

TYPE OF EQUIPMENT	SHIFTS BETWEEN OVERHAUL
Loaders	748
Shortwall cutters	1046
Universal cutters	1433
Locomotives	1187
Shuttle Cars	898

This figure is interesting in that it is exactly in the same ratio as for 1945. While there are minor variations in specific instances, the trend is extremely uniform throughout the industry.

The total number of loading machines represented by this report is in excess of 600 with corresponding numbers of cutting machines, locomotives, shuttle cars, and mine cars.

The committee feels that the cost data presented in this report should be valuable information for operating companies throughout the country. The committee will continue its work through 1947 and hopes that operating companies will supply them with information so that the work may be of value. The committee wishes to thank the mining companies who have so generously contributed data for this report.

Sectionalizing Power Distribution Underground

By A. Lee Barrett,* Member AIME

(Cincinnati Meeting, October 1947)

Mine power systems are quite different in many respects from those usually found in industrial plants. Wide areas are served, usually by a circuit which is connected continuously throughout the mine. In some mines one circuit may run for several miles. There are usually a small number of branch circuits involved considering the areas covered. In all mines circuits are sectionalized at approximately 2500 ft by means of switches. All, or at least most branch circuits, are sectionalized by manual switches. In some mines automatic sectionalizing gear is used to give automatic overload protection at various strategic points in the system. However at the present time this practice is by no means general.

Circuit hazards in mine electric systems are much greater than those encountered in industrial installation. Falls of roof or coal form a prominent hazard which may break the wire and may cause ground faults or partial ground faults. Ordinarily electric cables in mines cannot be insulated, therefore a fall can easily cause a short circuit.

The humidity in most mines is greater than 95 pct and at many points the floor is wet and water may be dripping from the roof over insulators, increasing the difficulties at that point.

All electrical circuits must be contained in the mine entries and space must also be provided for moving equipment and men. Consequently, because of the proximity of the power circuit to the locomotives and other moving equipment a considerable hazard is introduced. Mine haulage track is not always good so that wrecks are not unusual. A wreck frequently does considerable damage to the power circuit.

A ground fault frequently results in heavy arcing which may be sustained. If the damage to the circuit is mechanical it may mean that a trip of coal has wrecked and a great deal of coal dust could be thrown into the air as a result of the wreck. The arc resulting from the ground fault could ignite the coal dust and a mine explosion would result. A continued ground fault, even if high in resistance, also may ignite the coal which may be on the bottom of the entry and a mine fire may result. The fire hazard is one of the most serious hazards in any coal mine.

Since the mine circuit is completely interconnected, any power interruption resulting from a fault seriously affects the whole operation. Unless the mine is completely sectionalized the whole mine will be shut down until the fault is located and the proper switches are opened to isolate the fault; even then large areas may still be without power if they are supplied by a single power circuit which may be opened in isolating the fault. Further, mine power is usually supplied from a relatively small number of substations and the fault may be on a stub feed circuit, in which case power cannot be restored until the fault has been cleared.

Practice varies widely with respect to placement of substations. Usually they will be spaced from 1 to 2 miles in 500V operations, and at 1 mile intervals or less, in 250V operation. Actual mining operations govern substation placements and no general rules can be followed.

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* Pittsburgh Coal Company, Library, Pa.

As has been previously stated, sectionalizing overload practice and protection varies widely from mine to mine. It can be seen from the foregoing discussion that there is a great need for adequate overload protection of electrical circuits in mines. A great deal is yet to be done in this direction.

Installation and operation of overload protective and sectionalizing equipment should be handled with engineering care. Numerous instances have been brought to light where such equipment was serving no useful purpose because of circuit conditions and improper placement of the protective equipment.

The mine track system is used for one side of the electrical circuit in practically all mines; as a result circuit uncertainties are always present unless track bonding is checked from time to time.

A series of tests now being conducted by Mr. F. E. Griffith, formerly of the Pa. Dept. of Mines and currently with the U. S. Bureau of Mines, and Mr. C. L. Brown of the U. S. Bureau of Mines is uncovering some very interesting information with respect to mine circuits.

When published this information will undoubtedly be of great value to coal operators in laying out and operating mine electrical systems. Their tests show that mine circuit resistance approaches its theoretical value in few cases. Usually they are higher and it is not unusual for the resistance to be double the theoretical value. This points to the fact that more attention should be given to maintenance of mine circuits than is usually given and foreshadows a very pertinent problem in overload protection and sectionalization.

A number of cases has been found where the circuit resistance was high and the overload setting required to handle the normal load on the circuit was also high so that a direct short circuit, at the end of a stub circuit, would not result in enough current to trip the overload device. This is obviously a dangerous situation and one which should be corrected either with sectionalizing breakers or with adequate circuit resistance.

There are usually five points in the underground distribution system at which overload protection might be applied: (1) At the substation; (2) between substations; (3) at main branch circuit junctions; (4) at secondary branch circuit junctions; (5) at equipment connections.

At the substation protection is desired for the conversion equipment, for the station wiring, for the distribution circuit to sectionalizing breakers, and, in some instances, to the face. Usually the substation is located near the main haulage circuit and some protection is afforded main line locomotives by the overload protection provided in the substation.

Interrupting capacity required in overload equipment located at the substation is primarily determined by the capacity of the incoming alternating current line. Direct current circuit resistances in the proximity of substations are quite low and would allow excessive current to pass if not limited by incoming line characteristics. Usually an interrupting capacity of 25,000 amp is sufficient, though in some cases interrupting capacity of 50,000 amp is better practice.

Protection at this point usually consists of hand or automatic oil circuit breakers ahead of the conversion equipment and of individual direct current circuit breakers, either manual or automatic reclosing, for each direct current circuit leaving the substation. In some mines it is the practice to tie all circuits leaving the substation to one automatic circuit breaker. This lack of sectionalization results in circuit breaker settings minimizing circuit protection on individual circuits.

Feeder circuits almost always parallel and include the trolley wire, so that they are exposed to dripping water, falls, and wrecks occurring on the haulage ways. Fault resistances may vary from 100 ohms or more to 0 ohms in case the trolley wire directly contacts the rail. There is no sectionalizing equipment at the present time which is able to differentiate between a high resistance fault and a normal mine load. The normal instantaneous output of a mine substation is usually between 175 and 200 pct of the substation's current capacity. Depending on the circuit layout of the mine, all of this load or a part of it may appear on an individual distribution circuit. Obviously under these conditions it is very difficult for circuit breakers to provide much protection against mine fires resulting from high resistance ground faults.

The required circuit capacity of main feeder lines varies from mine to mine, and in a given mine. The usual overhead conductor capacity between substations is of the order of 1,000,000 cir mils including trolley wire and feeder wires in 500 volt mines, and 2,000,000 cir mils in 250 volt mines. The rail side of the circuit usually has approximately the same capacity, though in 500 volt mines with heavy track the capacity may be greater. Where automatic reclosing directs current, circuit breakers are used on the various feeder circuits out of the substation and attention should be given to the proper reclosing time. Considerable difficulties are encountered with equipment in the mine if the reclosing time is too short, while the loss of time would be too great if the reclosing time were long. A minimum reclosing time of approximately 20 sec is good practice and maximum reclosing time should not exceed one minute. Minimum reclosing time is established by the time required for motors to come to rest or be completely disconnected from the distribution circuit in such a manner that they will be recycled through their starting cycle when power is restored. If sufficient time is not provided, flashovers may occur on the motors involved.

Safe reclosing loads is also a matter for serious consideration at the substation. Good practice requires that permanently connected loads such as heaters and lights be held to a minimum as it is impossible for the breaker to differentiate between such loads and fault loads when attempting to reclose. It is poor practice to use any across-the-line starting motors controlled by knife switches on mine circuits, as the armature resistance of a direct current motor is very low at standstill.

On idle days and off-shifts it is common practice to shut down all but one or two substations at the mine, feeding all circuits from the remaining operating substations. This introduces an additional hazard because additional circuit resistance is inserted from a load point at the end of a mine circuit and the substation which may be serving that point. Under these circumstances circuit resistance can be so high that the circuit breaker will not trip even in case of a direct short circuit at the remote point. One method of eliminating this hazard is to provide two circuit breaker settings easily adjusted on the substation breakers—one setting for normal work-day conditions and a much lower one for idle days or off-shift conditions. This greatly improves protection against faults which might cause fires.

It is good practice to install an automatic reclosing sectionalizing circuit breaker in the main feeder lines between each two substations. Usually this tie breaker would be installed at the load center of the circuit running from one substation to the other. At this point the interchange of current on the circuit is a minimum and will allow for the lowest circuit breaker setting. This is an important consideration if several substations are used on a common circuit as it will assist in isolating the fault from a remote substation whose working breaker setting may be excessive when the distance from the fault to the substation is considered. Minimum circuit resistance between this tie breaker and the substation is usually in the order of 0.06 ohms for the 500 volt installation and 0.03 ohms in the

250 volt installation, so that interrupting capacities in excess of 10,000 amp are not required for this type of service. Maximum loads carried by this sectionalizing tie breaker will be less than half the maximum load fed to the circuit by substations at either end.

As with substation breakers, an idle day or idle shift presents certain problems. As in the case of the substation breaker, it is good practice to provide two current settings easily adjusted on the tie breaker so the current rating can be decreased considerably during idle periods. This will provide much better protection against high resistance faults which may not operate substation or tie breaker when operated at normal current settings.

Reclosing time of the tie breaker should be short as compared to the reclosing time of a substation breaker since the reclosing time of the tie breaker is added to the time of the substation breaker. If the tie breaker does not close in quickly there may be a serious increase in voltage on one side of the breaker on closing which could cause flashovers.

It is good practice to install sectionalizing circuit breakers at the junction of main circuits and interconnecting circuits between substations. A fair degree of protection may be obtained in 500V mines, however, without the use of main branch sectionalizers, providing circuit conductivity is good. In general main branch circuits feed face entries or panel entries which include the mine working sections. The average circuit resistance between the main branch sectionalizer and the substation would be in the order of 0.06 ohm for 500V mines and 0.03 ohm for 250V mines. These values are quite variable however since it is possible for a main branch circuit to break off very near a substation and since there is considerable variability in mine power systems. In 500V mines main branch circuits usually have about 500,000 cir mils in the copper circuit and the equivalent of about 1,000,000 cir mils in the track return. 250V mines in general have about 1,000,000 cir mils in each side of the circuit. Main branch circuits may extend to a distance of approximately 5000 ft. They are gradually built up from 0 as the section is developed to their full extent and gradually reapproach 0 as the section is worked out.

Interrupting capacities of 10,000 amp are usually adequate for these breakers though in certain instances where the circuit connects close to a substation higher interrupting capacity would be required. The power supplied to a main branch circuit is as variable as the extent of the panel entry or face entry. Breaker settings of around 1000 amp for 500V mines, and around 2000 amp for 250V mines are usual.

Reclosing time should be similar to that for substation breakers for the same reasons. Resistance of the main circuit at the time of reclosing should be as high as possible in order that good protection may be had against high resistance ground faults. Normally it would not be expected that there would be too many fixed loads such as lights or resistances in these areas.

A dual overload setting is good practice on sectionalizers main branch circuits, as they are useful on idle day and off-shift periods, when only a small pumping load may be carried. Much better protection is secured against fire if a dual overload relay setting is used.

Secondary branch circuits are usually working section circuits or butt entry circuits. In some instances two working sections may be a part of the secondary branch circuit. These circuits usually have a minimum length of approximately 2500 ft and build up from 0 to 2500 ft dropping back to 0 as the section is worked out. It is not general practice at the present time to use sectionalizing circuit breakers for secondary branch circuits; however it is thought that this would be a very good practice.

Secondary branch circuits may usually be adequately protected with 300–500 amp circuit breakers in 500V mines, and 600–1000 amp circuit breakers in 250V mines—settings are variable depending on the extent of the section.

Circuit resistance from the secondary branch circuit breaker to the substation is highly variable depending on the location of the junction of the secondary branch circuit with the main branch circuit. Good practice would indicate that in 500V mines the maximum resistance at the secondary branch circuit should be around 0.20 ohm and the minimum resistance to the secondary branch circuit breaker should be around 0.05 ohm; in 250V mines these values should be 0.10 ohm and 0.03 ohm approximately. These figures indicate that circuit breakers of 10,000 amp interrupting capacity should be used.

Various types of sectionalizing equipment may be used to protect secondary branch circuits. Automatic reclosing circuit breakers are probably the best overall but automatic reclosing contactors can be used satisfactorily as can hand reclosing circuit breakers. The difficulty with hand reclosing equipment is the time required for a man to travel from the working face to the secondary branch circuit breaker in order to reclose it and the lack of knowledge on the part of the operator as to what the circuit conditions are at the time of reclosure. It is entirely possible to use fuses for the protection of secondary branch circuits; however the same objections would apply to fuses as to hand reclosing circuit breakers and the cost of operation would be higher. Reclosing time for secondary branch circuit breakers should be approximately 20 sec in order to give equipment time to clear the line.

The best idle day protection for secondary branch circuits is to disconnect the power; however if it is necessary to maintain power secondary branch circuits dual overload relay elements can be used giving the maximum protection during times of low load. A common practice with considerable merit is to provide small fuses for these sections during idle time, which can be inserted or cut out by the operation of knife switches.

Electrical equipment used in mines can be classed in three general groups; stationary equipment, such as pumps, compressors and shop equipment; semiportable equipment, such as belt, shaker and chain conveyors; elevating conveyors and car moving equipment and portable equipment, such as locomotives, mining machines, shuttle cars, loading machines and drills. Portable equipment is usually supplied with cable reels and lengths of cable running from 350 to 500 ft.

Usually stationary equipment is installed in a manner very similar to surface stationary equipment. Generally the wiring is protected adequately and danger from fire is reasonably remote. Circuit hazards too are greatly reduced as stationary equipment is usually located in well timbered areas or in underground buildings of tile, steel, concrete block, or brick. Choice of protective equipment depends primarily on the size of the equipment. In most instances fuses are adequate and most economical.

Semiportable equipment such as conveyors are usually wired in a less protected manner so that good circuit protection is desirable to insure protection against fire or explosion. Frequently conveyors are so located and ventilated that coal dust may introduce a hazard. While this can be kept definitely under control by the use of water and rock dust, it is a sound plan to add all the protection in the circuit that is feasible.

The best protection for this equipment and its wiring is an automatic or more usually a manual overload circuit breaker. Contactors are also satisfactory since an interrupting capacity of approximately 5000 amp is adequate because of the high circuit resistance from the equipment to the substation. Secondary branch circuits feeding this equipment are usually approximately 211,000 cir mils in 500V mines and 422,000 cir mils for 250V mines. On the basis of average circuit conditions thruout the mine, it would be expected that the average resistance from stationary equipment to the substation would be in the order of 0.15 ohm for 500V mines and 0.075 ohm in 250V mines. Resistance from the equipment to the substation can, however, vary widely, depending on the location of the equipment on the secondary by-circuit, the location of the secondary branch circuit on the main branch circuit, and the location of the main branch circuit on the main feed lines. Fuses present the difficulty that the blowing time of larger size fuses is long. This may mean an arc from several seconds to as much as a minute duration before the circuit is open. Obviously this introduces unwanted hazards in dusty areas.

Portable equipment, such as shuttle cars, locomotives, cutting machines, and loading machines are equipped with trailing cables ranging in length from 350–500 ft. 500V mines in general use No. 6 and No. 4 cables, whereas 250V mines use No. 4, 2 and 1 cables, depending on the size of the equipment involved. The following table indicates the circuit resistance of various cable sizes in common used.

TABLE 1—*Circuit Resistance of Various Cable Sizes*

CABLE SIZE	OHMS RESISTANCE FOR 500-FT CABLE
6	0.40
4	0.25
2	0.16
1	0.13
0	0.07

It will be noticed that these cables add a significant amount of resistance to the circuit on which they are used and that the current flowing in case of a fault in the cable is reasonably well limited.

Portable cables are subject to a number of hazards including falls of roof, mechanical damage in handling, and possible damage from associated equipment. They present on failure one of the most serious problems in the electrical circuit since they are in the active working place where gas and dust accumulations are most likely to exist.

Most portable cables are presently sectionalized by fuses at the point where they attach to the main power circuit of the mine. In order to hold trolley tap cable fuses to small sizes, fuses are manufactured which are not approved by the Bureau of Underwriters because of their small size. As a result there is considerable variability in the blowing time of fuses manufactured by different concerns. When failures occur it is not unusual for a cable arc to persist (under certain conditions) for as much as one minute when fuses are used.

It is thought that very much better protection could be had from the use of circuit breakers at this point since fuses require considerable time before blowing in case of a fault. Experimental installations of small circuit breakers in trolley taps have indicated that a greatly improved situation exists when they are used. It is reported when such circuit breakers are used no flashing has been seen when cables fail. At this time there is need for the development of a small trolley tap circuit breaker which would be light in weight—preferably less than 15 lb—rugged in construction, so that it could be handled rather roughly in moving the equipment, and reasonable in cost—preferably not more than $50.00.

It is realized that the considerable distances involved in mine electrical circuits make adequate servicing, testing, and inspection a problem. However the hazards present are serious. It is considered that a great deal more than ordinary engineering study should be given to the layout and design of electric circuits underground. Overload protection and sectionalization should be carefully planned to insure that the mine is properly protected under all conditions. Inspection could be improved profitably in order to prevent the wide divergence of theoretical and actual circuit resistance in the mine. Such a program would go far toward improving the economics of power consumption in mining as well as toward increasing the safety of the mining operation.

Methods of Disposal and Handling of Refuse at Anthracite Mines in Eastern Pennsylvania

By George J. Clark,* Member AIME

One of the major problems of operation in the anthracite industry is the disposal and handling of refuse—not because of its complexity but because of the quantity and type of material involved. It is the purpose of this paper to describe the kinds of refuse, methods of disposal, quantities of material to be handled and equipment used, together with present day application at specific operations in the Northern, Eastern Middle, Western Middle and Southern anthracite fields.

For the purpose of this paper, refuse is considered to be any material that has no present or foreseeable future value and that must be handled and disposed of in the process of mining and preparing anthracite for market. Included in this classification are tunnel rock, mine rock, cleaner plant refuse, breaker refuse, old timber, ashes and silt.

Disposal Methods

Old timber is disposed of either by burning or by depositing on the refuse bank. Burning is preferred because timber on the refuse bank is a potential fire hazard.

Ashes are deposited on a separate bank, mixed with other refuse, or flushed into old mine workings. In many places, the latter practice has caused considerable difficulty in preparation of coal mined from these areas. In depositing ashes on the surface, it is wise to refrain from mixing them with material containing coal as such contamination may prevent reclamation of the coal in future years.

Tunnel work, mine rock, cleaner plant and breaker refuse compose the bulk of the material to be disposed of and handled, and are referred to collectively as "refuse" in this paper. Waste material in or adjacent to the coal beds may be removed at the working face wherever possible. This can be done in flat or lightly pitching beds, and in the Northern field represents a sizable percentage of this type of refuse handled. In the other three coal fields, this method of disposal cannot often be used. Refuse not removed at the working face is taken to the surface and generally is deposited on banks; occasionally it is crushed and flushed into old mine workings. Breaker refuse often is used for constructing berms for silt-retention basins.

Disposal Problems

The problem of handling refuse is different for each operation, owing to topographical conditions, quantity of material to be handled, anticipated requirements and types of equipment in use elsewhere at the operation. Preparation plants with limited refuse-disposal areas, excessive grades to reach the required elevation and long hauls to disposal areas not readily accessible require the selection of equipment to best meet the conditions. The quantity of refuse to be handled from a given amount of raw material varies considerably throughout the four fields. In mine coal, this varies from a low

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percentage in the flat-pitching beds, where a comparatively clean car of coal is obtained by gobbing the refuse, to a high percentage in pitching measures where all material, including the extraneous material adjacent to the bed, is loaded by gravity into the cars and taken to the surface. The quantity of refuse also varies with the kind and quality of the raw feed. An operation handling a high percentage of raw material from strippings in heavily mined pitching beds or from low-grade banks will be obliged to handle a relatively high percentage of refuse from a given amount of raw feed.

Table 1.—*Relative Amounts of Refuse Handled on the Surface, Exclusive of Silt, in the Four Anthracite Fields during* 1945

Field	Quantity of Refuse Handled Expressed as Percentage of Raw Material by Weight				
	Straight Rock	Refuse in Coal			
		Removed at Primary Cleaning Plants	Removed at Breaker	Total Refuse in Coal	Total Refuse Handled on Surface
Northern	6.8	0.2	16.4	16.6	23.4
Eastern Middle	6.7	15.5	28.2	43.7	50.4
Western Middle	2.5	10.6	35.2	45.8	48.3
Southern	2.6	3.4	41.1	44.5	47.1

The relative amount of refuse handled on the surface in each of the four anthracite fields during 1945 (Table 1) varies from 23.4 pct of total raw material for the Northern field to 50.4 pct for the Eastern Middle field. The amount of refuse is expressed as a percentage by weight of the total raw material handled. All types of raw coal feeds are bulked and all refuse included, except silt and ashes, whether handled directly as straight rock, removed at a primary cleaning plant, or removed at the breaker. These averages are based on the experience of companies producing approximately 65 pct of the anthracite in eastern Pennsylvania, and are considered representative of each field.

Refuse-handling Equipment

In the selection of refuse-handling equipment, consideration should also be given to the anticipated life of the operation and to possible further use or salvage value of equipment. Types of equipment in use elsewhere in the work should be considered; for example, at a stripping operation trucks used for hauling coal or overburden can be used part time to handle cleaner-plant refuse.

Refuse is handled on the surface by many types of equipment, and also to haul the waste material to the refuse bank. The commonest types of equipment are mine cars, refuse cars, trains, incline cars, belt conveyors, belt stackers, flight conveyors, aerial tramways and trucks.

At some operations, particularly in the Northern field, straight rock, which consists of material not put through the preparation plant, is taken directly to the dumping point in mine cars.

Cars for hauling refuse in the anthracite field have been manufactured by a number of companies and have been made and used extensively in a wide variety of sizes and styles. Cars have been built in sizes ranging from capacities of about 100 cu ft to 38 cu yd in the end-dump, side-dump or three-way dump types, with stationary bodies and sloping bottoms, rocker-type bodies and flat-bottom drop-side bodies. The cars are operated in trains drawn by steam or electric locomotives, or are equipped with direct-current electric motors, which enable them to operate as individual units. These motorized cars are designed to operate on limiting grades of from 3 to 10 pct and at speeds ranging from 5 to 10 miles per hour.

Motorized trains consisting of a load-carrying locomotive and one to three cars are in use in all four anthracite fields. The trains generally are designed to operate on limiting grades and speeds similar to those for the motorized refuse cars. It is essential for one of the cars in the train to be equipped with an end-dump body, to aid in extending the dump and in building and maintaining the uniformly graded track required for efficient operation. It is believed that the first train with a load-carrying locomotive was put in use in the anthracite field by the Susquehanna Collieries Co. at its Pennsylvania colliery near Mt. Carmel, about 1929, and was used to haul raw coal to a storage pile.

Belt conveyors, in widths from 24 to 48 in., are used extensively for handling refuse. Some carry the refuse directly to the disposal point, where it is distributed over the bank by stackers; other belts elevate the refuse to a hopper and from which it is hauled to the disposal point by means of cars or trains. Belts are operated on grades up to 16° or slightly more if conditions demand, and at speeds up to 550 fpm. Belt installations are favored at operations where a large quantity of refuse must be handled over a long period of time, where the refuse must be raised to a considerable height to provide disposal area, and where obstacles must be cleared to reach the disposal site. In designing and specifying belt installations, careful consideration must be given to the maximum size and quantity of material to be handled.

Flight conveyors have long been used for handling certain kinds of refuse, generally for gathering or elevating refuse to a pocket for further handling. However, at several plants in the Middle Western and Southern fields, flight conveyors have a major part in the disposal systems. (One of these is described later in this paper.) In general, the flight conveyors are suitable for operating on grades up to about 30° and at speeds to about 150 ft per min. These steep operating grades permit deposition of material in relatively inaccessible areas and in high banks where space is limited.

The aerial tramway is not commonly used but has the advantage of a long suspended span for overhead crossing of existing improvements or other obstacles and for reaching remote disposal areas. (An aerial tramway installation at the Huber colliery of the Glen Alden Coal Co. is described later in this paper.)

In some installations, refuse is elevated by means of incline cars, which may be used to deposit material directly on the bank or into hoppers for further handling by cars, trains or stackers.

During the past 10 years, the use of gasoline or diesel-powered trucks, in sizes from 5 to 15 cu yd capacity, has become increasingly popular for hauling refuse. Their advantages are principally an extreme flexibility of operation, small initial investment, and usual immediate availability. In general, trucks are used as the principal method of hauling refuse at operations where one or more of the following conditions exist: a short haul, a small amount of material, a relatively short life, small or isolated disposal site or sites that cannot be reached economically by other means, or where the trucks can be used part time to handle refuse and the remaining time on other hauling. Trucks are used generally where part of the refuse is hauled to a separate location for building berms for silt-disposal basins or for backfilling.

Some coal companies let contracts for hauling their refuse.

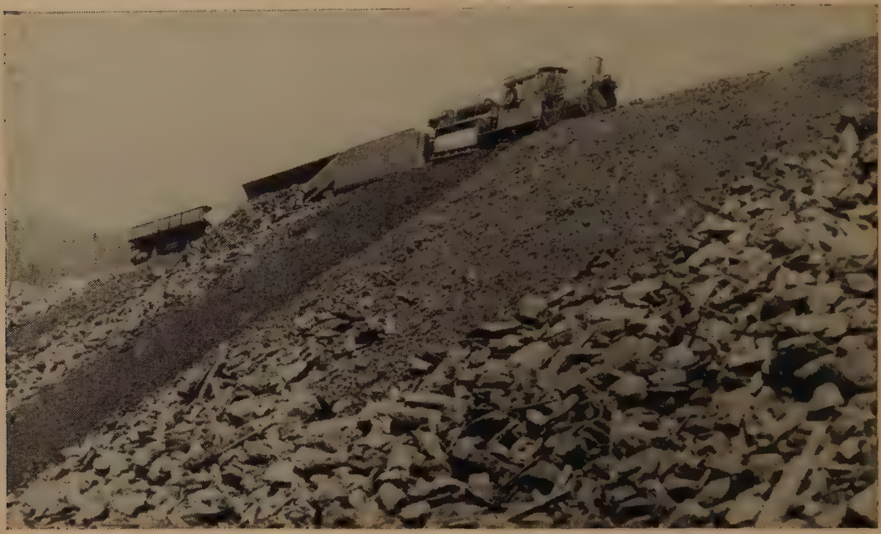

FIG 1—REFUSE BANK AT TAMAQUA COLLIERY.

Present-day application of some of the various types of refuse-handling equipment is given in the following descriptions of operations throughout the four fields.

TAMAQUA COLLIERY, LEHIGH NAVIGATION COAL COMPANY

Refuse at the Tamaqua breaker of the Lehigh Navigation Coal Co., in the Southern field, at Tamaqua, is handled by standard-gauge railroad equipment. During 1945, 937,000 net tons of refuse were handled by this system at a rate of about 250 tons per hour.

Two trains, each having a 73-ton locomotive, 67-ton tender and two 30-cu yd side-dump cars, haul the refuse from the pockets at breaker up a 3-pct grade railroad (80-lb rail) to the dump site, which is at present a haul of 6800 ft to the end of bank. The cars of the train are tilted to a dumping position and righted by pistons powered by compressed air generated by the locomotive.

The refuse throughout the two breaker shifts is dumped along the slope or face of the bank (Fig 1), which has a length of 1500 ft and a height of about 75 ft. At the end of the day's dumping, the material that gradually builds up on the dumping side of the track is dressed by the spreader car, which is pushed by one of the refuse locomotives.

On the front of the spreader car are heavy plow blades (Fig 2), which can be elevated, and on each side of the car there are hinged plow blades that can be elevated and swung to the position desired while dressing the bank along the track. The blades are powered by compressed air generated by the locomotive.

The rail on the dump side of the track is held within 7 ft of the top edge of the bank, to facilitate the dumping of cars. A track shifter (Fig 3) eliminates some of the heavy hand work formerly required in shifting and grading track. A section of track is shifted by securing the clamps of the shifter car to the tracks, then the heavy spud rack is planted between the rails in the desirable position for shifting. A mechanism driven by gas motor elevates the shifter car along the spud rack, which in turn raises and thrusts the track to the alignment desired. Each operation of the track shifter moves the track about half the width of gauge, except on curves, where this distance is not so great.

The location and elevation of the bank dictated the use of a back switch on this system. Two turnouts are provided, with an electric signal system, which reduce transportation delays.

GLEN LYON COLLIERY, SUSQUEHANNA COLLIERIES COMPANY

Refuse at the Glen Lyon colliery of the Susquehanna Collieries Co., in the Northern field, near Nanticoke, is handled by a combination of two balanced incline cars or gunboats and a motorized train. During 1945, this system handled an average of 650 net tons and a high day of 1100 net tons of refuse in a 7-hr period.

Breaker refuse and platform rock, after crushing, are carried by scraper conveyor to a hopper for loading into the gunboat. Straight rock is dumped directly into the gunboat from the mine car.

The gunboats are hauled up a double-tracked 22° incline, 800 ft long, by a double-drum hoist driven by a 175-hp, 2200-volt motor. The gunboats are about 250-cu ft capacity and make a trip in approximately two minutes.

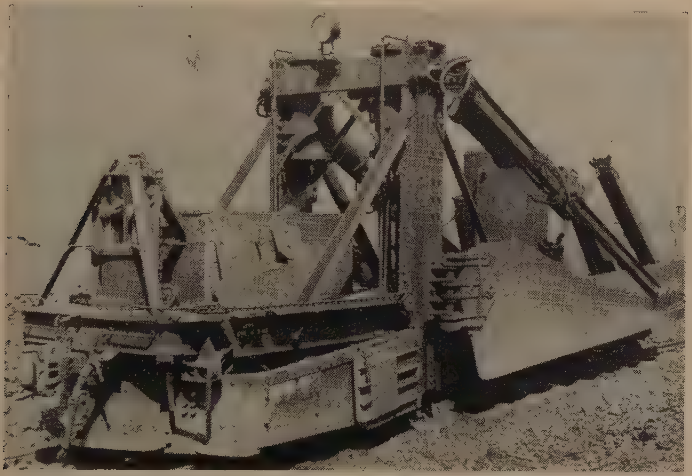

FIG 2—SPREADER CAR AT TAMAQUA COLLIERY.

Refuse is dumped automatically from the gunboats into a hopper for transfer into the motorized train. The hopper is equipped with an electrically driven gate, which is operated by the motorman from the cab of the train.

The train consists of a control car of 18 cu yd capacity and a 15-cu yd three-way dump car (Fig 4). The cars are dumped by air from the control car. The train is driven by two 100-hp, 250-volt direct-current motors mounted on the control car. Direct current is furnished by a 300-kw motor-generator set. The train is equipped with dynamic and air braking systems.

It is designed to operate at 10 miles per hour on a maximum grade of 8.25 pct. The present standard-gauge track, constructed of 50-lb rail, is approximately 2000 ft long, part of which is on an 8.25 pct grade.

A bulldozer is used to spread the refuse on the bank, to eliminate excessive moving of the track.

ST. NICHOLAS BREAKER, PHILADELPHIA AND READING COAL AND IRON COMPANY

The St. Nicholas breaker of The Philadelphia and Reading Coal and Iron Co., in the Western Middle field, near Mahanoy City, is the preparation plant for coal mined in the Shenandoah, Mahanoy City and Tamaqua areas. Refuse at this breaker is handled by a combination of belt conveyors and trains. The raw coal entering the breaker is rough-cleaned at the outlying sources; that is, all large refuse is removed and the remaining material is crushed to 6-in. size or under. Consequently, the refuse can be handled satisfactorily on belts without additional crushing. During 1945 this system handled a total of 1,398,000 net tons of refuse with a maximum of 7118 net tons and an average of 6019 net tons of refuse in a 7-hr period.

Refuse from the primary cones flows by gravity to the main belt. Other refuse is collected, after desanding and dewatering, on a 42-in. belt conveyor, which discharges onto the main refuse belt. The gathering conveyor is 182 ft long, constructed on a maximum grade of 15°48′. The belt is driven by a 40-hp, 440-volt motor and operates at a speed of approximately 520 fpm.

The original main refuse belt conveys the refuse from the breaker to a 120-ton hopper located on the refuse bank. From the hopper the refuse is discharged into

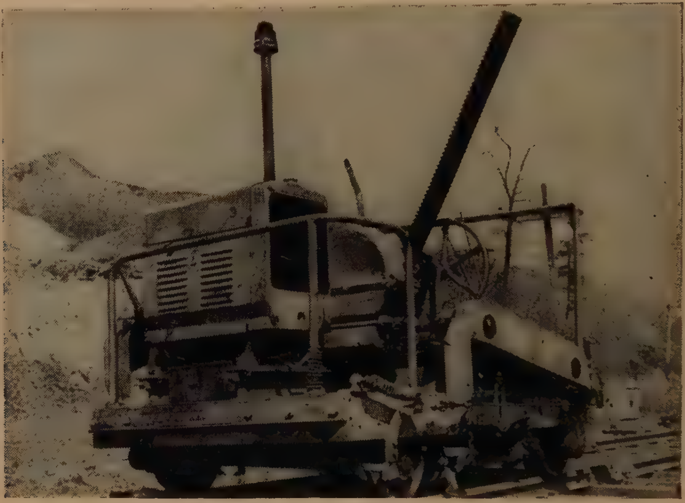

FIG 3—TRACK SHIFTER.

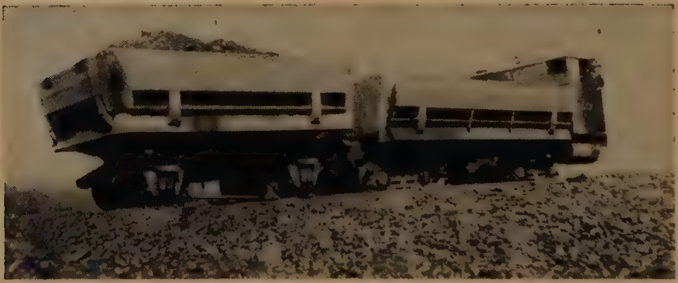

FIG 4—MOTORIZED TRAIN.

trains on a track level approximately 126 ft above the collecting point at the breaker. The dumping space, within an economical haul distance, on this original level, is nearly exhausted and for this reason a second leg, including a 180-ton hopper, was added to the conveyor in 1945, to raise the refuse approximately 123 ft vertically to a new train operating level. At present, part of the refuse is handled by the second leg and is used to construct a fill at the head end of the conveyor for the new level. Fig 5 shows a general view of the conveyor and Fig 6 shows progress of work on new level.

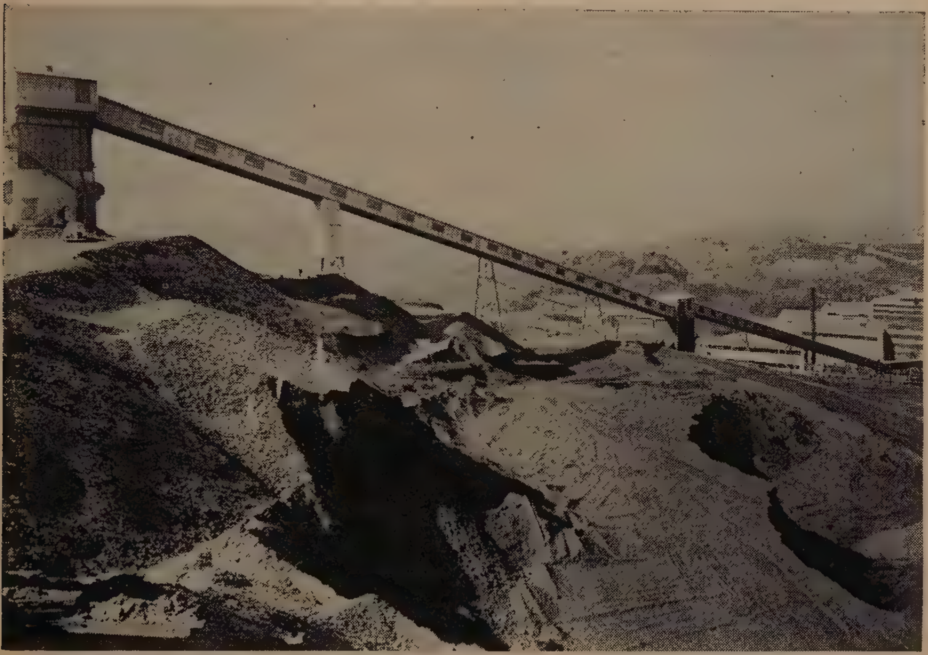

FIG 5—CONVEYOR AT ST. NICHOLAS BREAKER.

The main refuse belt is 48 in. wide and is operated at a speed of 500 fpm. The first leg of the conveyor is 837 ft long on a grade of 14°17′; the second leg is 442 ft long, on a grade of 16°15′. Each is equipped with a 250-hp 2300-volt motor.

Refuse is hauled from the hopper to the disposal point by two trains. The trains were designed for operation on 3 pct continuous grade, with a maximum capacity of 123 cu yd made up of a 32-cu yd capacity locomotive, two 38-cu yd two-way side-dump cars and a 15-cu yd end-dump car. Different combinations of cars are used to suit varying circumstances. Each train is driven by four 105-hp, 250-volt direct-current motors, all mounted on the locomotive, and is equipped with air brakes and air dumping mechanism. Direct

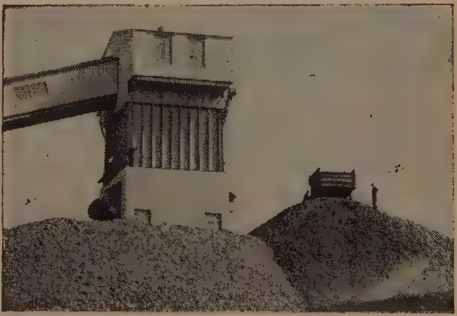

FIG 6—PROGRESS OF WORK ON NEW LEVEL OF REFUSE BANK, ST. NICHOLAS BREAKER.

current for operating the equipment is furnished at 250 volts by two 300-kw motor-generator sets.

There is approximately 9000 ft of standard-gauge track, constructed of 100-lb rail, on the present operating level of the refuse bank. Four dumping points are provided at haul distances varying from 200 to 4400 ft. The front end section of the body on the end-dump car when in the dumping position is used as a plow for spreading material for building and grading track.

The train is shown in loading position at 120-ton hopper in Fig 7 and in dumping position in Fig 8.

Oak Hill Colliery, Philadelphia and Reading Coal and Iron Company

At the Oak Hill colliery of The Philadelphia and Reading Coal and Iron Co., at Minersville, Pennsylvania, in the Southern field, a four-car train was put in operation on April 19, 1945, to handle refuse, replacing a motorized lorry-car system. By use of the train, it will be possible to deposit an additional layer of refuse at a maximum height of 200 ft over the old refuse bank. The maximum haul of 4000 ft is slightly less than that of the old lorry system. From the start of operation to Jan. 1, 1946, approximately 2520 tons of refuse was handled daily by the train in 14 hr, with a high day of 3000 tons. The estimated capacity of the train is 3500 tons at a haul distance of 4000 ft, and is ample to meet the refuse-handling requirements of the colliery.

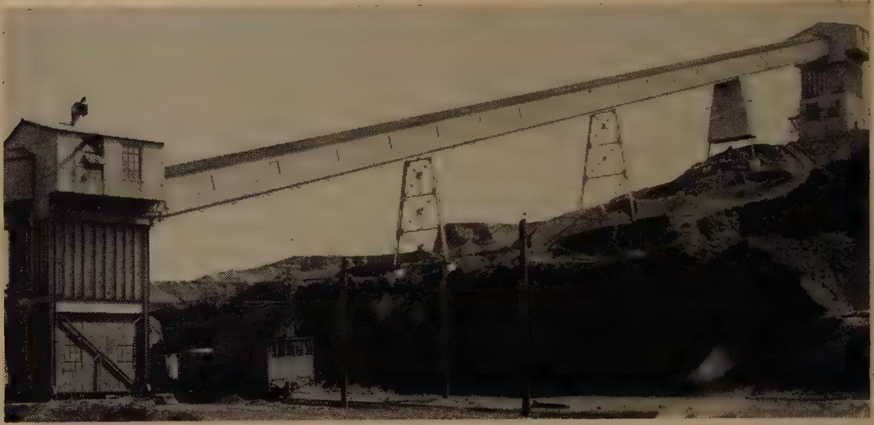

Fig 7—Train in loading position, St. Nicholas breaker.

Oak Hill breaker prepares the raw coal from Oak Hill mine, and the material from several outlying sources. Some of the outside feeds are raw coal and some rough-cleaned coal, the product of primary cleaning plants. The large rock from the raw coal is hand-picked and flows by gravity to a rock chute. The refuse from the various preparation facilities is collected on a 24-in. scraper conveyor and elevated to slate pockets. Straight rock and old timber are dumped directly from the mine cars into a chute that is outside of the breaker. All refuse is loaded directly into the refuse train from the chutes (Fig 9).

The train consists of a motorized control car two motorized trailer cars and an end-dump trailer car with water-level capacities of 16, 20, 20, and 15 cu yd, respectively. The tare weight of the train is approximately 71 tons, and the gross weight, fully loaded, about 161 tons. The motorized cars have two-way side-dump bodies and the trailer car a two-way side and end-dump body (Fig 10). The dumping mechanism is air operated and the cars can be dumped either individually or in a group from the control car or individually at each car.

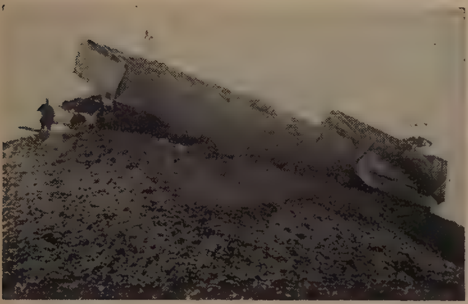

Fig 8—Train in dumping position, St. Nicholas refuse bank.

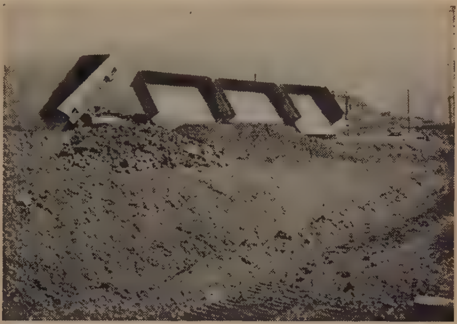

Fig 10—Cars in dumping position, Oak Hill refuse bank.

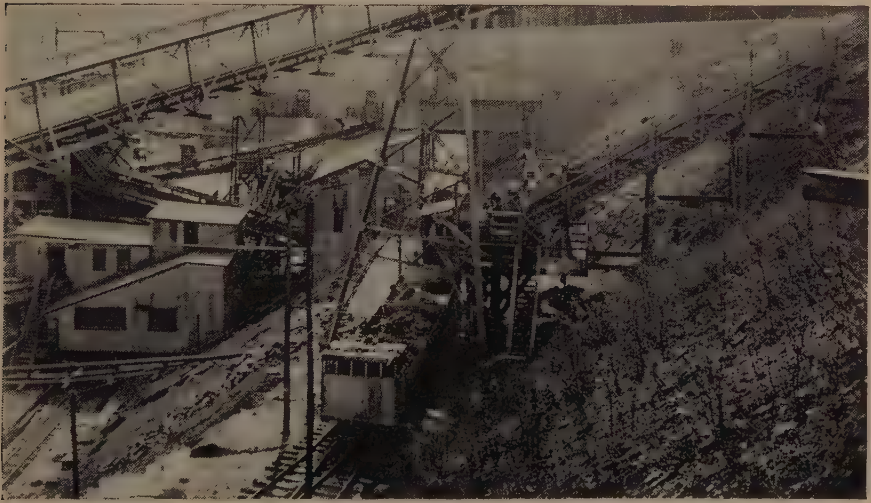

Fig 9—Train being loaded from chute, Oak Hill colliery.

The train is driven by six 126-hp 600-volt direct-current motors, two on the control car and two on each of the motorized trailer cars. All are operated from the control car. Direct current to operate the train is furnished by a 750-kw, 600-volt sealed-tube ignitron rectifier. The train is equipped with dynamic and air braking systems.

The train is designed to operate at 10 miles per hour on a 10 pct grade. The first leg of the track is approximately 1900 ft long and contains level-loading and back-switching sections and approximately 1350 ft on 10 pct grade with vertical curve approaches. The second leg advancing at present is about 1300 ft long, virtually all on 8 pct grade. A bulldozer is used to help spread the refuse on the bank. The first leg and back switch are constructed of 100-lb rail; the second leg of 70-lb rail.

Fig 11 shows a train climbing the 10 pct section of track.

Maple Hill Cleaner Plant, Philadelphia and Reading Coal and Iron Company

At the Maple Hill cleaner plant of The Philadelphia and Reading Coal and Iron Co., in the Western Middle field, near Mahanoy City, refuse is handled by a combination of belt conveyor and motorized side-dump refuse cars. This plant serves as a primary cleaner for raw material from several sources furnishing coal for the St. Nicholas breaker. During 1945, this operation handled a maximum of 2400 net tons and an average of 1660 net tons of refuse in a 14-hr period.

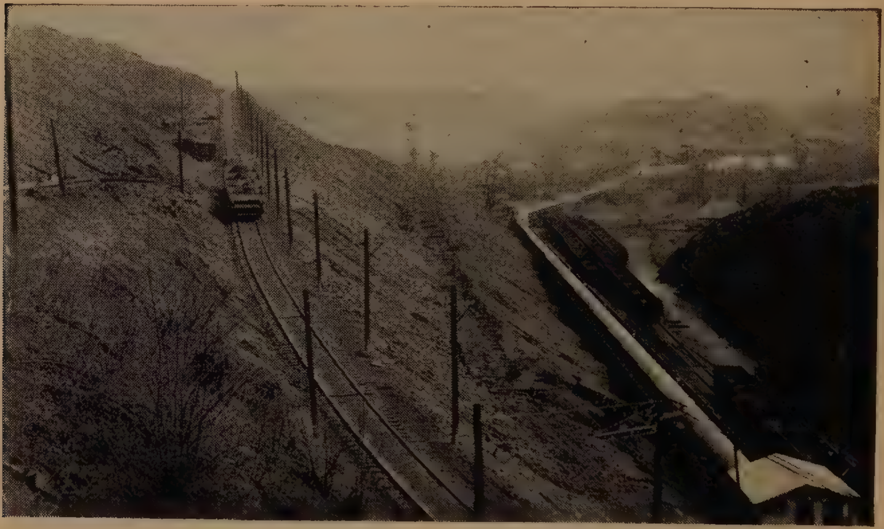

Fig 11—Train climbing 10 per cent grade, Oak Hill refuse bank.

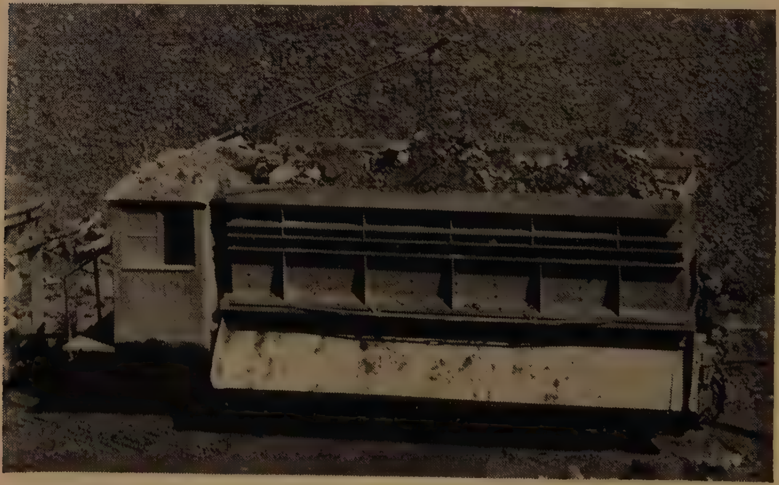

Fig 12—Side-dump refuse car, Maple Hill cleaner plant.

The plus 12-in. refuse in the raw coal is removed by hand at picking tables and then conveyed to a 36-in. gyratory crusher, which breaks the rock into 6-in. or smaller sizes. Straight rock is fed directly into the

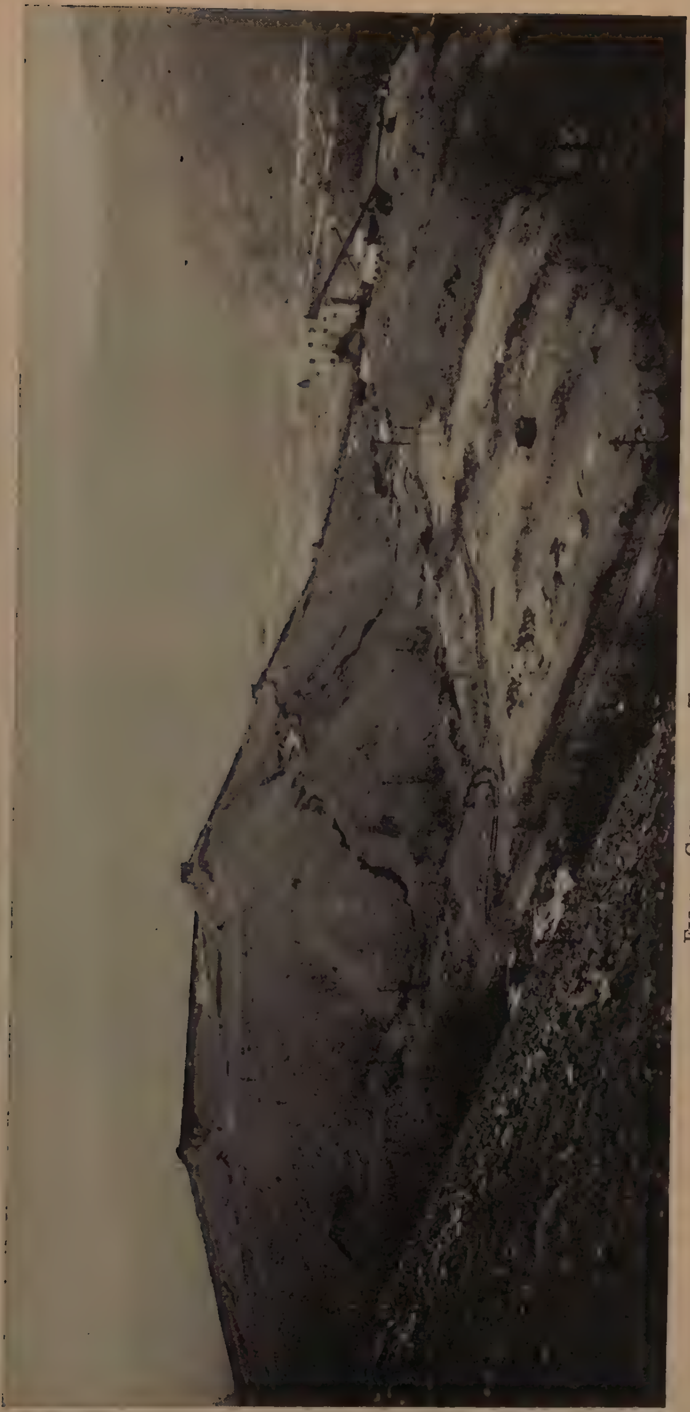

Fig 13—Conveyors at Trevorton breaker.

crusher, which is driven by a 150-hp motor. The crusher product and hand-picked minus 12-in. refuse is transported on a 48-in. belt conveyor to an 80-ton bin on the refuse bank, from which it is discharged into side-dump refuse cars (Fig 12) on a track level approximately 162 ft vertically above the collecting point at the cleaner plant.

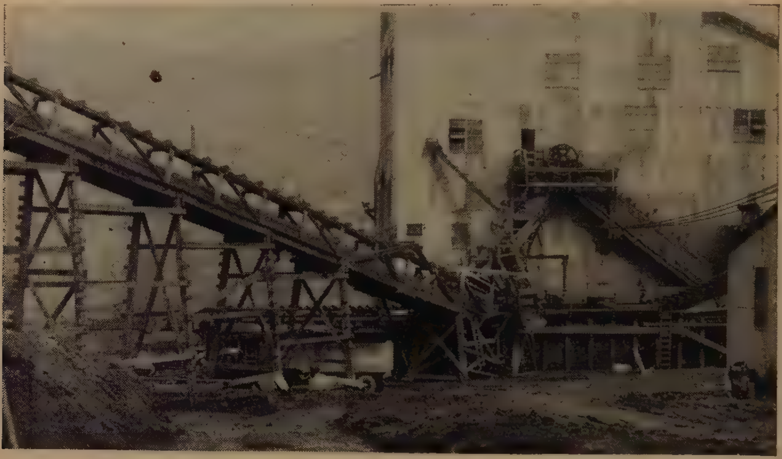

FIG 14—GENERAL CONSTRUCTION OF CONVEYOR AT BREAKER END, TREVORTON.

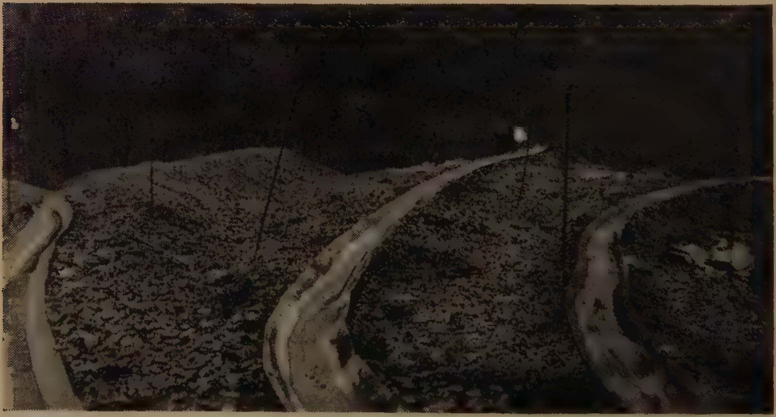

FIG 15—TROUGHS FOR WASHING REFUSE ONTO BANK, TREVORTON BREAKER.

The conveyor is approximately 941 ft long, is constructed on an average grade of 12°49′, and is driven by a 300-hp, 2300-volt motor at a speed of approximately 550 fpm.

The crusher and belt installation in this instance was adopted to eliminate the operation of a refuse train or lorry cars, which, on 3 pct grade, would require a distance of approximately one mile to reach the same elevation.

Two 20-ton motorized side-dump refuse cars haul the refuse from the bin approximately 700 ft to the present dumping site. Each car is equipped with two 50-hp, 250-volt direct-current motors. The cars are designed to operate on grades up to 3 pct. The track is 44-in. gauge, 60-lb rail, and is on a plus 1.4 pct grade.

TREVORTON BREAKER, STEVENS COAL COMPANY

Refuse at the Trevorton breaker of the Stevens Coal Co., in the Western Middle field, at Trevorton, is handled by a combination of flight and belt conveyors (Fig 13). During 1945, more than 1,000,000 net tons of refuse was handled at a rate of about 190 tons per hour.

The system consists of three legs of flight conveyor and one leg of belt conveyor. Flight conveyors were selected for handling refuse because of the limited area available for dumping, which made it necessary to go to a great height in the shortest distance. Since the conveyor was put into operation in 1942, the bank has reached a vertical height of 215 feet.

Flights are 10 by 27 in. spaced on 36-in. centers and travel at a speed of 150 fpm. The trough is wood lined with cast-iron pans; bents are of wood (Figs 14 and 15).

Conveyor legs are 200 ft long on a rise of 3½ in. in 12 in. (16°16′); 240 ft on a rise of 6 in. in 12 in. (26°34′); and 210 ft on a rise of 6 in. in 12 in. (26°34′). Conveyors are driven by 50, 75 and 75-hp, 440-volt motors, respectively.

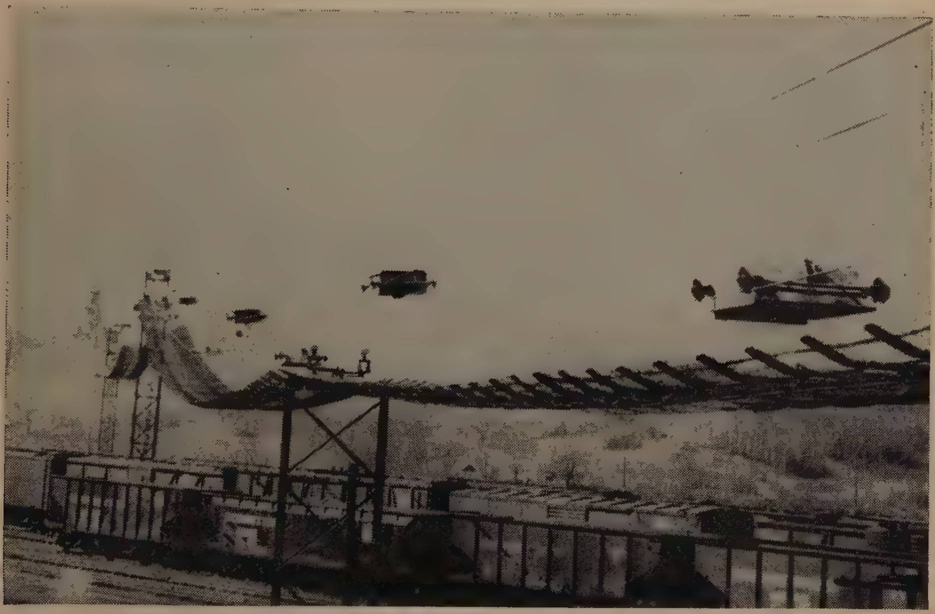

FIG 16—CONTINUOUS AERIAL TRAMWAY, HUBER COLLIERY.

The belt conveyor is 300 ft long and is operated in a horizontal position on the top of the bank, to spread the material over a wider area. The 24-in. belt is driven by a 15-hp motor at 450 fpm.

To obtain a further spread of the material, the belt discharges on sheet iron, on which, with the aid of a stream of water, the refuse is carried a greater lateral distance. Approximately 300 gpm of water is used for washing the refuse.

HUBER COLLIERY, GLEN ALDEN COAL COMPANY

At the Huber colliery of The Glen Alden Coal Co., in the Northern field at Ashley, breaker refuse is handled by an aerial tramway system (Fig 16). During 1945 the tramway handled an average of

1150 net tons in a 7-hr period, which is close to the capacity of the installation.

Breaker refuse is conveyed on a belt to the tramway loading terminal, where it is delivered into a 10-ton steel bin. From here the refuse flows by gravity to a loading hopper of 20 cu ft capacity, mounted on a counter-weighted cradle and equipped with stops and limit switches for automatic operation. From this hopper, the refuse is loaded into 20-cu ft cars, mounted on a continuous type aerial tramway, which transports it to a 100-ton capacity bin at a transfer terminal a distance of approximately 1650 ft. Twenty-seven cars are attached to the continuous haul rope, which is driven at a speed of 400 fpm by a 25-hp motor. The load-carrying cables are of two pieces, 1⅛-in. diameter, special construction lock-coil track cable. The empty return cables are two pieces, ¾-in. diameter, special construction. The haul rope is ⅝-in. diameter plow-steel hoisting rope.

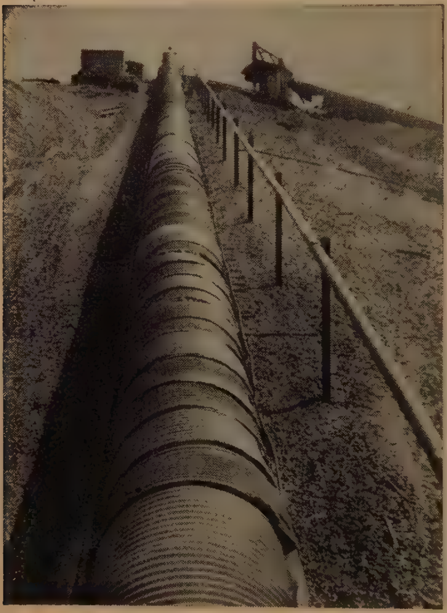

FIG 17—CONVEYOR NO. 3, MARVINE COLLIERY.

The refuse is transferred from the 100-ton bin at the transfer terminal to a 100-cu ft traverse hopper and from the hopper to two V-shaped tramcars of 100-cu ft capacity, each operating on a separate disposal line. Each of the two diverging disposal lines is 1750 ft long. An intermediate or breakover tower, 150 ft high, is set in each line at a distance of 410 ft from the terminal, leaving a net disposal length of 1340 ft on each line between break-over tower and tail tower. The tail towers are 250 ft high. The tram cars are operated at a speed of 900 fpm by means of a haul rope, continuous from terminal through tail towers, and driven by a 100-hp motor. The tramcar door is opened automatically at a predetermined distance from the loading point. The haul cable is ¾-in. diameter, plow-steel grade. The cars travel on two 1⅜-in. diameter track cables, special lock-coil construction.

MARVINE COLLIERY, HUDSON COAL COMPANY

At the Hudson Coal Company's Marvine colliery breaker, at Scranton, in the Northern field, breaker refuse is transported by means of a series of belt conveyors to the disposal bank. The final refuse, which is of nut size and smaller, is deposited on the belt of conveyor No. 1 at the end of the breaker.

Conveyor No. 1 is a 30-in. belt conveyor, 402.0 ft long, traveling at the rate of 200 fpm, driven by 20-hp motor, snubbed drive, and is mostly contained within a concrete tunnel that conducts the conveyor under the tracks of the New York, Ontario and Western R.R. The conveyor is on level grade for the first 150 ft and then rises on a long-radius, vertical curve for 224 ft, to a maximum rising grade of 4⅜ in. in 12 in. at the headhouse, where it deposits onto Conveyor No. 2.

Conveyor No. 2 is a 30-in. belt, 325.0 ft long, traveling at the rate of 230 fpm

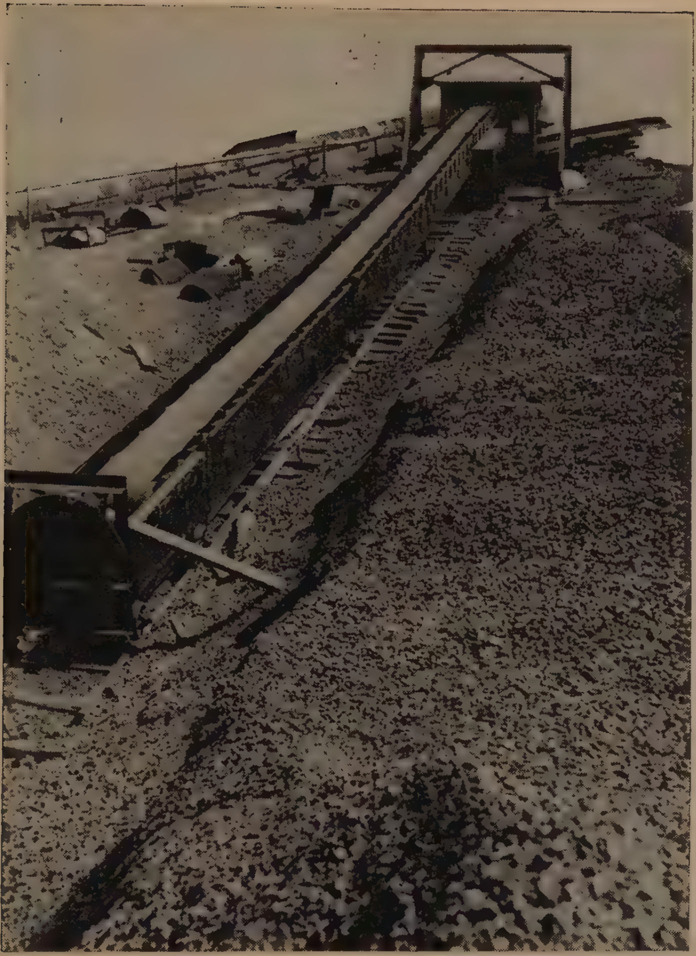

Fig 18—Conveyor No. 4, Marvine colliery.

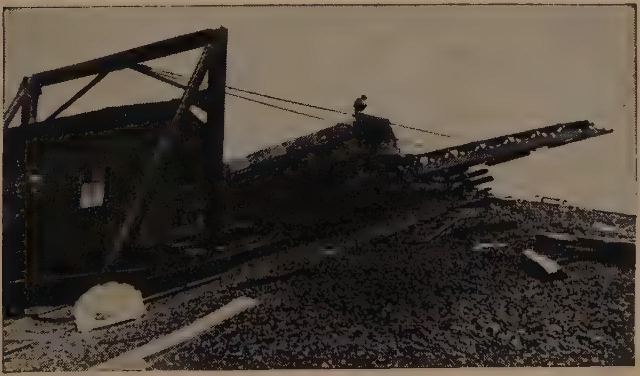

Fig 19—Conveyor No. 5, Marvine colliery.

and driven by a 30-hp motor, tandem drive. This section of the conveyor is constructed overhead on an upgrade of 3⅝ in. in 12 in. It is contained within a steel-truss, metal-covered housing, which carries the conveyor over Boulevard Avenue (a busy highway) to a headhouse on the side of the refuse bank, where it deposits the refuse on the shuttle conveyor, or conveyor No. 3.

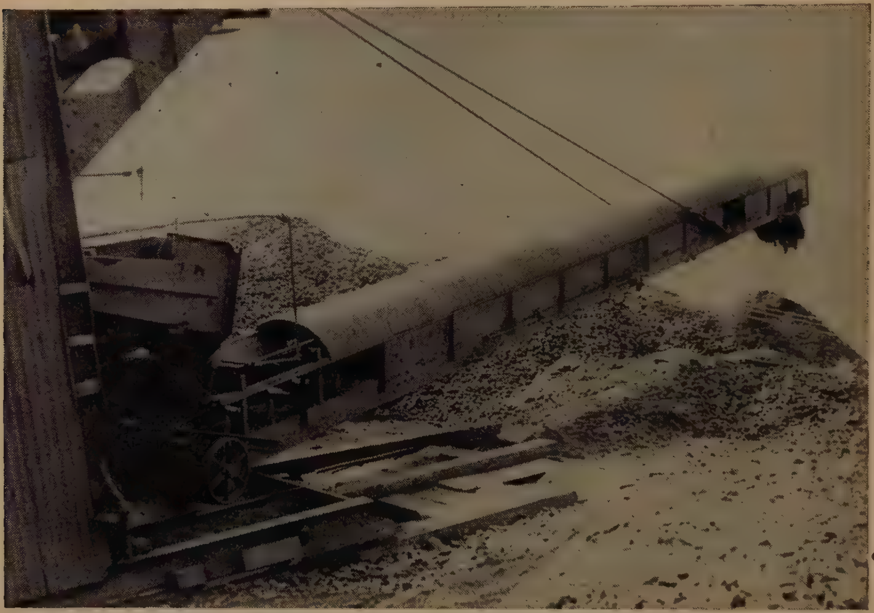

FIG 20—SWINGING-BOOM CONVEYOR AT LOREE COLLIERY.

Conveyor No. 3 (Fig 17) is at present a fully extended shuttle conveyor. It is a 30-in belt conveyor, 325.0 ft long, traveling at the rate of 260 fpm on a rising grade of 3⅝ in. in 12 in. This shuttle conveyor was originally 200 ft long and was a completely assembled belt conveyor suspended under permanent conveyor No. 2. As dumping proceeded, and the bank formed in front of the conveyor, the shuttle was advanced on mine-car wheels, attached to the conveyor frame, and these, in turn, were supported by a track of 90-lb rail on 4-ft 3-in. gauge laid on the top slope of the bank. The conveyor was moved forward by means of a hoist and series of pulleys anchored to the 90-lb rail track. Since the shuttle was placed in operation in 1932, several intermediate sections have been added to extend the dumping facilities. These sections are 36 ft long and were connected to the other sections of the conveyor by a hinged joint. The hinged joints allow for uneven settlement of the bank.

Conveyor No. 4 (Fig 18) is an assembled section of another self-contained shuttle conveyor (No. 4) that is to be installed at the head of conveyor No. 3 at an angle of 60° to the present conveyor line. This shuttle is a 30-in. belt conveyor, 145.0 ft long, and will be driven at the rate of 300 fpm by a 30-hp motor, snubbed drive. The conveyor will be supported and advanced on a 90-lb rail track rising on a grade of 3⅝ in. in 12 in. It is designed so that complete sections may be added until the final maximum length of 285 ft is obtained. Conveyor No. 4 will discharge to conveyor No. 5.

Conveyor No. 5 (Fig 19) is a 30-in. swinging-boom belt conveyor, 50.0 ft long, capable of swinging through an arc of 80° on each side of the centerline, and will be driven at the rate of 300 fpm by a 7½-hp. motor, plain drive. The swinging-boom conveyor will be connected to conveyor No. 4 by a flexible joint and both will be moved forward by a hoist installed within the frame of conveyor No. 4.

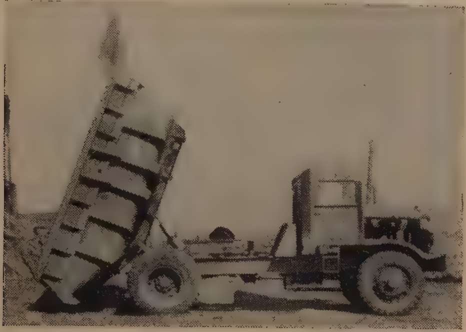
Fig 21—Type of truck used at Hazleton Shaft colliery.

The design capacity of the equipment is 200 tons per hour. In 1945 an average of 165 tons per hour was handled and deposited on the refuse bank.

The shuttle conveyors and swinging-boom conveyor are protected from the weather by a semicircular housing of 16-gauge corrugated steel. In cold weather heat is carried under the housing by means of steam pipes.

Fig 20 shows a 50-ft swinging-boom conveyor in use at The Hudson Coal Company's Loree colliery at Plymouth, Pa. The boom operation is typical of what will be in place at Marvine colliery at the head of the conveyor lines.

Hazleton Shaft Colliery, Lehigh Valley Coal Company

Refuse at the Hazleton shaft colliery of the Lehigh Valley Coal Co., in the Eastern Middle coal field at Hazleton, is hauled by a combination of trucks and a locomotive-drawn refuse car.

Refuse obtained from rough-cleaning bank material is hauled a distance of 900 ft from the cleaning plant to the refuse bank by two 15-cu-yd diesel-powered end-dump trucks. The two trucks haul an average of 120 loads of refuse in a 14-hr period.

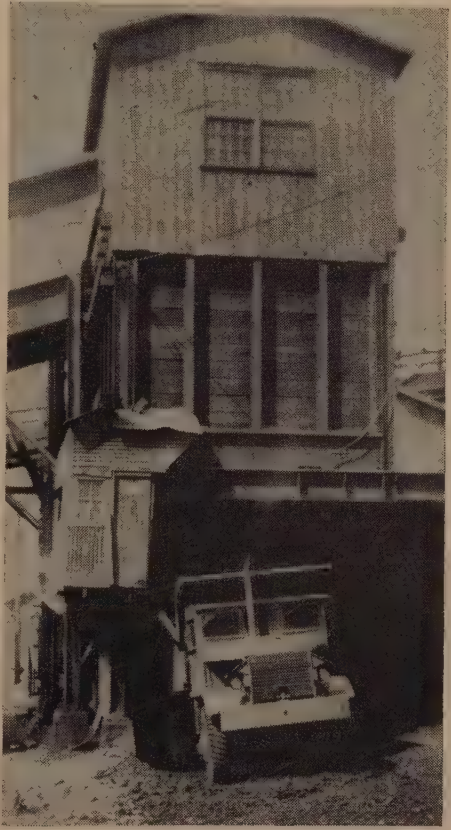
Fig 22—Truck at loading chute, Hazleton Shaft colliery.

Breaker refuse is hauled by four 15-cu-yd trucks (Fig 21) from a hopper at the breaker up a 5 pct grade approximately 2350 ft to the present disposal point. Each truck averages 45 trips in a 14-hr period.

Straight rock dumped from the mine cars is by-passed to a rock chute by means of a power-operated gate. From the chute, the rock is loaded into a 30-cu-yd refuse car (Fig 22) and then hauled 2000 ft to the rock bank by a 40-ton standard-gauge steam locomotive.

Before trucks were used for hauling the breaker refuse, the system consisted of a gunboat on a plane and refuse cars drawn by a steam locomotive. The trucking of refuse was decided upon to permit the reclaiming of a bank on which the other facilities were located.

ACKNOWLEDGMENTS

For assistance in obtaining material and preparing this report, the author wishes to thank and give credit to C. D. Rubert, Superintendent of Preparation, Lehigh Navigation Coal Co.; H. A. Dierks, Mining Engineer, Glen Alden Coal Co.; E. T. Powell, Mining Engineer, Susquehanna Collieries Co.; George Holland, Assistant Mining Engineer, Jeddo-Highland Coal Co.; H. H. Otto, Assistant General Manager in Charge of Engineering, The Hudson Coal Co.; H. W. Montz, Mining Engineer, The Lehigh Valley Coal Co.; W. C. Muehlhof, Chief Engineer, The Philadelphia and Reading Coal and Iron Co.; and other members of the Engineering Department of The Philadelphia and Reading Coal and Iron Company.

Experience with a Training Program

By J. E. Norton,* Member AIME

(Chicago Meeting, February 1946)

Production statistics show that during the period of emergence from the depression the coal industry was becoming increasingly cognizant of the economic and competitive necessity for mechanizing. About 1935 many of its leaders were becoming aware also of the alarming scarcity of men properly trained to cope with the new problems of mechanization. In order to get a factual statement of this problem, a special AIME Coal Division Committee† was requested to survey the condition. Its investigation disclosed that graduates of mining schools going into coal mining numbered 16 in 1937 and 15 in 1938.

With this as a background, Pittsburgh Coal Co. foresaw that a comprehensive program was necessary if it was to enlist the services of trained technical men, who were an indispensable part of a mechanization program. The purpose of this paper is to review the program set up, to comment on the successes and failures and to outline the future course of action.

Summer Work

An inescapable conclusion from the mechanization statistics was that the opportunities in coal mining were good. Not only was there a void in technical skill to fill from past years, but the accelerating rate of mechanization indicated an increasing demand for technically trained men for the future. The remedy lay primarily in presenting facts concerning the coal industry to the undergraduate engineers before they made their professional occupational choice. Based on this idea, one of the key points in the program was summer vacation work for the undergraduate, which, through the inducement of summer earnings, brought men into the mines to take a look at the prospects. This was more successful than statistics indicate because the war scrambled badly the orderly working of the program, but it could not obscure the value of a summer work program. A decided advantage was that it gave the company a preview of the graduate. An observation on this part of the program is that it is extremely hard to gauge lower classmen, and we found it wise to give them the benefit of the doubt, as the last two years in college bring about surprisingly rapid development in some men. We could readily single out the men at the opposite ends of the scale of competence.

Comprehensive Training Program

Having adopted summer work as a means of interesting undergraduates, the problem of attracting graduates was met by establishing a training program that covered approximately 120 weeks and

Table 1.—*Training Schedule*

Location	Weeks
Introduction	1
Underground mining operations	27
Coal preparation	24
Inspection—safety, mech., prep.	31
Mine engineering and survey	16
Industrial relations	12
Operating staff—costs, time study	8
Miscellaneous	1
Total	120

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† N. G. Alford, *Chairman;* C. T. Hayden, E. A. Holbrook, C. E. Lawall and L. E. Young.

put the men into all the main parts of the Production Division. Table 1 gives an outline of the schedule.

The first step in the program, whether for graduate or undergraduate, was a trip to all our mines, plants, shop and staff departments to get acquainted. The second step was always underground mining. Here the men worked on track, timbering, supplies handling, on a face crew, and other operations. This experience is basic, of course, and for many will be their only intimate first-hand experience of working as one of a crew of men on a job. The remainder of the schedule was followed in general; the duration and sequence varied somewhat but seldom did the men fail to put in approximately the prescribed time in each department.

TABLE 2.—*Number of Technical Men Employed*[a]

Year	Undergraduates (Summer Work)	Graduates
1938	1	1
1939	12	10
1940	33	13
1941	34	12
1942	22	5
1943	4	6
1944	1	1
1945		
Total	107	48

[a] Thirteen of the Summer Work Undergraduates joined the company on graduation. 29 of them spent more than one summer, so the net total number of different men was 113.

TABLE 3.—*Schools and Number of Men*

SCHOOL	NUMBER OF MEN
Pennsylvania State College	27
Pittsburgh	15
Colorado School of Mines	14
Ohio State	11
Lafayette	10
Missouri School of Mines	8
Alabama	8
Lehigh	5
Virginia Polytechnic	4
Illinois	2
Columbia	1
Southern California	1
Colorado	1
Washington	1
Pennsylvania	1
Grove City	1
Lincoln University	1
Marshall	1
Wisconsin Institute of Technology	1
Total	113

TABLE 4.—*Present Status of Graduates Employed*

PRESENT STATUS	NUMBER OF MEN
Armed Services	12
Resignations	13
Unknown	1
Mine supervision	2
Operating staff—costs, time study	10
Engineering	2
Preparation	4
Maintenance	1
Inspection (safety)	1
General inside labor	2
Total	48

The Industrial Relations Department was charged with the responsibility for scheduling the men through the various operations and departments as well as for following their personal progress. It consulted frequently with them and encouraged them to express their ideas frankly.

Tables 2, 3 and 4 show the number of men in the program, the different schools represented and the present status of graduates employed.

CONCLUSIONS

As Table 2 shows, we have had experience with 113 undergraduate and graduate engineers and we are prepared to inventory our program. Because of the gap in mining engineering matriculants due to the war, and the uncertainty of those who are still in the armed forces, our future plans are unsettled in respect to some details, but it is possible to examine their broad outlines in the light of our past experience.

In the first place, we are satisfied that the industry needs technically trained men, and we must continue to offer them inducements to attract them; in other words, we will have to compete for our share of the men. We believe both summer work and an established training program are inducements, and we plan to follow the same general training program but expect it to differ somewhat in details from the past for reasons subsequently given.

A formal planned training program not only serves to attract a man; it orients him in his industry and educates him in the most practical way possible by introducing him to all the different phases of the business. Since, in the nature of things, he gravitates toward specialized work—maintenance, operations, engineering, preparation—he will never have an opportunity to see all the other phases unless we consciously and deliberately route him through them. A further reason is that since he has been introduced to all the main parts of the business, the likelihood is increased that he will enter the specialization phase in a type of work congenial to his temperament and ability.

Our program scheduled wage increases every 6 months for 2 years—provided, of course, the man continued to show promise. If he did not, he was either dropped or told he was on trial and the reasons explained to him. This wage policy, we believe, is a proper one. A schedule of wage increases tended to bring his earnings up faster than other employees and we often encountered the condition where an inexperienced graduate got more pay on a job than an experienced incumbent, but this problem has to be met according to the circumstances, because, after all, there are few inducements as compelling as wages, and we have to meet the competition for men from other industries.

DIFFICULTIES ENCOUNTERED

The operation of a training program in itself tends to establish in the graduate's mind a realization that he is not lost in the shuffle but close observation and attention are desirable both from the point of assuring the man we are interested in him and of enabling the company to follow his progress. But it is on this part of our program that we have encountered some of our most exasperating problems. Some are inevitable, some were caused by our inexperience, and some we attribute at least in part to the colleges and universities. A discussion of some of the difficulties from each class follows. The discussion is necessarily generalized and does not fit all cases.

Practical necessity puts a time limitation on any training course. A better nomenclature would be "induction course," since a two-year period involving experience in six or eight major branches can be little more than introductory. There was a disposition on the part of some of our men to feel they knew a branch when they had merely observed its workings, and they became impatient to move along to the next branch. Consequently, they failed to appreciate the opportunity that was being presented to them. Since the age bracket of these men is 22 to 26 years, this may be tantamount to a criticism of their youth.

Another difficulty inherent in our program was the need for moving periodically. This is often genuinely a disturbing influence in their personal life. There is a succession of housing, boarding and transportation difficulties as well as recurring readjustments in their social life, and in the case of those either contemplating or embracing a marital status, the personal inconvenience is serious.

Conditioning Men for Responsibility

As might be expected, the inauguration of a large-scale program coupled with our lack of first-hand experience led us into a few troubles. At least one of these was quite serious but is nevertheless rectifiable. It was our failure to assign direct job responsibilities to the men during the training period. Naturally our permanent personnel felt their everyday work had to come first, and the education of the transitory apprentice was often viewed as a gratuitous chore. Inasmuch as the key personnel in a department were the ones who had the knowledge we wished the trainee to acquire, it was often a heavy drain on the time and energies of our best men and to some degree diverted them from their duties. The facile solutions, and ones we unwittingly followed too often, were either to treat the learner as an observer without assigned tasks or responsibilities or else to assign him to routine work that required a modicum of instruction and attention from the mentor. There is no simple remedy for this trouble. It requires a definite scheme of training that assigns jobs and the responsibility for their completion within each department, and we were remiss in failing to set up an intradepartment plan with the same care we used in setting up our overall program. Our program suffered somewhat because too few men are good instructors and some coaching along this line seems called for in the future.

A slightly different aspect of this was our failure to condition the men to accept responsibility. The Industrial Relations Department scheduled the various moves for the men and had frequent consultations with them. The men knew their activities were controlled out of that department and came to feel that they were responsible almost solely to it. This feeling, coupled with the failure to assign work to them that was of real consequence, brought about something of a floater complex. It is pertinent to observe here that the trainees were aware of this and it was frequently a subject of discussion among themselves as well as with us.

One way to meet this difficulty would be to assign a man to one mine or operation and to use that as a pivotal point instead of the Industrial Relations Department. For example, a trainee, after completing his underground service, would do cost work for the superintendent for a period, then go into the Engineering Department ment with an assigned task but would be held over for a period sufficient to cover a prescribed course. This would be repeated in the Maintenance Department, Preparation, and so on, but he would always go back to the mine for a period to maintain a psychological association with a fixed place. It should not be assumed that this plan would cause him to forego any part of the training schedule. The Industrial Relations Department would plan the moves, have consultations and keep close check as at present, but the trainee would be administratively responsible to a line official rather than to the Industrial Relations Department.

There are difficulties of another sort for which the responsibility rests jointly with ourselves and the colleges and universities. These stem from the failure to condition the men mentally for the long, hard grind they face before reaching a position of responsibility. The competition for the services of graduates has distorted their perspective. Regardless of his competency, probably it will be at least five years before a graduate fills a job carrying true responsibility or where his managerial or engineering talents are given any free rein. Our men seem to expect to attain an industrial status commensurate with their four-year college degree after two years in industry. From our point of view, only their induction is completed at this stage and there is another three to five years of application ahead before their learning has had enough tempering by experience to form judgment. Between us, we must condition them to the cold, hard fact that, while the prospects in coal mining are excellent, they face a trial period, involving both time and hard work.

In what has been said heretofore there has been undue emphasis on our troubles because we are trying to think our way out of them. The next few years will pay dividends on our efforts, as we can see already in some of the boys the benefits of our investment in training.

COLLEGE AND UNIVERSITY TRAINING

Before discussing the training received by the men in colleges and universities before they come to us, we refer back to the second paragraph on page 1, in which the services of technically trained men are called an "indispensable accessory to a mechanization program." This is and will continue to be true. Our purpose in bringing attention to this statement is to make certain that it is not lost sight of despite the meanderings into other fields.

A good starting point for this discussion is an analysis of our requirements for men as evidenced by the kinds of jobs we have and the educational background that would be most helpful. Table 4, showing the present status of our younger men, is of some interest, but it covers too short and too hectic a span of years to be conclusive.

In order to make something of a factual approach to our requirements for men, an annual salary of approximately $4000 was arbitrarily selected, and all jobs in the Production Division paying that amount or more were classified into four groups. More than 200 jobs were included. First the Engineering Group occupations were picked out, and then the remainder were sorted into the other three groups. A definition of the various groups is:

1. Engineering. Includes jobs requiring a high degree of specialized technical knowledge. Examples: Chief Engineer, Preparation, Mining Engineer, Maintenance Engineer.

2. Supervisory. Includes supervisory and managerial skill. Examples: Production Manager, Purchasing Agent, Mine Supervisor.

3. Advisory Staff. Includes nonsupervisory, nonengineering experience. Examples: Safety Inspector, Mine Inspector.

4. Practical Experience. Includes jobs capable of being filled by nontechnical experienced men; some are supervisory. Examples: Master Mechanic, Mechanical Inspector.

Table 5 shows the results of such a classification. Note that there is a reasonable correlation with Table 4, despite the limited experience represented in the latter.

TABLE 5.—*Classification of Occupations Paying Approximately* $4000 *or More per Year*

OCCUPATIONAL GROUP, PRODUCTION DIVISION	PERCENTAGE OF TOTAL
1. Engineering	9
2. Supervisory	79
3. Advisory Staff	4
4. Practical Experience	8
Total	100

The last two classes in Table 5 are occupations that can be satisfactorily filled by intelligent men with the proper experience and aptitude, and do not require formal education beyond the high school level, so are not particularly relevant to this discussion. The other two groups are. Note that there are eight or nine supervisory jobs for each engineering job. This is certainly an expression of the ratio of opportunities between the two classes of occupations, although it may leave unanswered other very pertinent questions, such as how many men are in the supervisory group because they have good technical background.

There can be no doubt whatsoever of our need for a high level of technical competency in the engineering group and it is a rare situation indeed when any company finds itself with an engineering department that is well balanced in all the necessary branches. Likewise, there can be no doubt that there will continue to be a strong demand for this class of graduates from the colleges.

REQUIREMENTS FOR SUPERVISORY WORK

Turning to the numerically predominant Supervisory Group, it is proper to ask the requirements for these jobs. What degree of technical training is necessary?

What degree of social skill is necessary? What degree of managerial skill? Granted there will not be the same answer in every case, some sound generalizations are possible. One certainly is that we need all three.

With the important exception of those in the Engineering Group, practically all of our technical graduates receive more specialized engineering training than they will ever be called on to use, and, since our experience records indicate that the opportunities are in the Supervisory Group, good men naturally gravitate toward the opportunities. In supervisory work, a sound technical background is extremely helpful and we cannot do without enough good engineering talent scattered among our Supervision Group to solve certain kinds of on-the-job problems that are truly in the engineer's province.

On the other hand, we are becoming increasingly apprehensive of our lack of social skill, which we can define roughly in this connection as the ability of the supervisory arm of management to enlist the willing cooperation of our working force. With no intention of disparaging the use or discounting the value of time study and wage-incentive plans, which put job performance and earnings into the empirical, hence the engineering sphere, some recent studies, of which the one at the Hawthorne plant of Western Electric,* conducted by Harvard University, is a good example, are bringing individual and group psychology factors to light that may have more significance to industry than Taylor's studies. Some of our liberal industrial leaders are convinced that the next important step in the attainment of better levels of production will come as a result of a better understanding and application of the social and psychological factors that motivate the working force and cause them to cooperate spontaneously as members of a production team.

The New Deal gave impetus to the unionization of labor, and union leadership unfortunately has striven to establish in the minds of its membership that management is their natural enemy and the union is their friend and protector. The strong mutuality of interest has been played down, and, although there are always more points of reciprocal interest than of conflict, management must find the means for sympathetic communication of such identical interests. Elton Mayo, Professor of Industrial Research at Harvard University, in "The Social Problems of an Industrial Civilization," says, "If our social skills had advanced step by step with our technical skills, there would not have been another European war." Some pessimistic thinkers are fearful lest the lag between social and technical progress become so great that revolution instead of evolution will result.

The case for managerial skill—the art of manipulating the available tools and forces to yield a result—is too obvious to need much elaboration. The need for attaining low costs to survive competition and for profits to satisfy the owner is fundamental in a free-enterprise economy, and some basic grounding in such economic facts of life is indispensable.

Engineering and Nonengineering Types

We have indicated above that the majority of our opportunities are in the Supervisory Group, and that the talents required are engineering, social and managerial in whatever varying proportions. Since our training program has been almost exclusively for mining-school men, we come to the question: Do these men have the balanced background we require? An observation not susceptible to formal proof but strong enough to be a conviction is that many men, by virtue of their personality—used in the broadest sense to cover aptitudes, leadership, intelligence, academic competence, introversion-extroversion factors, and so on—tend to classify into engineering and nonengineering types.

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* F. J. Roethlisberger and W. J. Dickson: Management and the Worker. Harvard Univ. Press, 1939.

Could not faculty members make a reasonably good forecast, say at the end of two years, of those who would make good engineers and those who would not? The good engineering prospect would be encouraged to continue in engineering work to graduation and perhaps to graduate work. The other group, already grounded in fundamentals of mathematics, physics, chemistry, geology, and so on, would continue only such engineering courses as are generally helpful. The remainder of their courses would be those that would be helpful to them as prospective supervisors. For example, in the social science field, they would study individual and behavior psychology, to learn why we behave like human beings. They would take sociology to enable them to understand better the origins and organization of our society and the study of masses of people. They would get more history—American, English and labor—from which to obtain an understanding of our traditions and economics; and accounting, to get the feel of our system.

In "Human Leadership in Industry," Sam A. Lewisohn* expresses his thoughts on this subject:

> The question is: what preparation have they had to act as such a buffer? A background limited to physics, chemistry, mathematics, mechanics, and other specific sciences does not equip a man to act as a "buffer between labor and capital." Some engineer executives who are absorbed and effective in the technical problems of management neglect those which may be termed the "political" aspects.
>
> It is especially significant that managers are so largely products of the technical schools and that foremen too are now being recruited from this source. For if engineers are so frequently nominated to management, not only the technical problems but also the management of the human relations at the various plants are in their hands. It is futile to attempt to make progress in problems of human relations without their cooperation.
>
> It is a question of creating among managers new patterns of social behavior and approach. Aside from reaping the advantage of profits which spring from good management, employers should boast of having secured the good will of their workers as they do of their other successes.

Prof. Elton Mayo's conclusions are much the same, although arrived at from an entirely different approach. Our present day society has failed to give the worker security such as he had in the less dynamic industrial life of a few decades ago. The Hawthorne experiments of Mayo's associates found that increased group performance followed attention—the individual members of the group became a team and produced better, simply because interest was being shown in them. Once the feeling of "being on a team" was engendered, their stability and security seemed to be restored to them and a spontaneous and freely offered increase in productivity followed. This is not an isolated laboratory phenomenon, since we have all seen what happens when any gang pitches in whole-heartedly to do a job. Mr. Lewisohn points out it is the responsibility of new leadership to understand and apply the principles. We feel that a more balanced technical, social and economic education is the next and necessary step in that direction.

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* Harper and Brothers, 1945.

The Use of Smoke Apparatus in Practical Mine Ventilation Work

By A. T. Beckwith,* Member AIME

(New York Meeting, March 1947)

This paper concerns measurements of low-velocity air currents and investigations on mine ventilation by means of chemical smoke. The chemical smoke used is produced without flame and at ordinary mine temperatures, and therefore presents no ignition hazard in gaseous mines.

Mining men who have worked in open-light mines are familiar with the use of tobacco smoke in determining the direction of air travel and approximately measuring low-velocity air currents. The use of chemical smoke is merely a refinement on the use of tobacco smoke and the flameless low-temperature source of chemical smoke has the additional advantage that it can be used in gaseous mines.

Ventilation smoke apparatus, Fig 1, is a simple device for producing dense white smoke. The apparatus as shown consists of a rubber aspirator bulb which is used to force air through a glass "smoke tube" containing a chemical substance that produces a dense white smoke on contact with air. The apparatus was developed by engineers of the U. S. Bureau of Mines.[1] It is inexpensive, weighs less than 3 oz, is easily carried in a pocket, and requires no skill to operate.

Smoke apparatus is invaluable in practical mine ventilation work for: (1) measuring low-velocity air currents, (2) detecting air leakage, (3) observing air-flow characteristics.

This paper is written to show the merits of the apparatus, and is based upon findings through its extensive use, during the past six years, in mines operated by the Lehigh Navigation Coal Co. Inc.

The Apparatus

Certain chemicals, such as stannic chloride ($SnCl_4$) and pyrosulphuric acid ($H_2S_2O_7$) produce "white smoke" when exposed to air. Stannic chloride gives off vapors which on contact with moist air cause the formation of solid hydrates, of which $SnCl_4 \cdot 5H_2O$ is the most important. Pyrosulphuric acid decomposes when exposed to air with the evolution of sulphur trioxide (SO_3) fumes, which become white upon contacting moisture in the air.

A chemical, such as one of the above, is the active substance in a smoke tube. The glass tubes are hermetically sealed during manufacture, and are made ready for use by merely breaking off the tips at the conveniently located file cuts. When the smoke tube is connected to the aspirator, depressing the aspirator bulb forces air into contact with the chemical, and smoke discharges from the open end of the tube.

A white solid (a hydrate if stannic chloride is used, or sulphur trioxide if pyrosulphuric acid is used) may occasionally form a cake in the outlet opening of a smoke tube. A small wire such as an ordinary paper clip can be conveniently employed to dislodge the cake should it seal the tube.

When the assembled apparatus is not

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* Special Engineer, Lehigh Navigation Coal Co. Inc., Lansford, Pennsylvania.

[1] References are at the end of the paper.

in actual use, a rubber cap, such as a detachable pencil eraser, should be placed over the exposed end of the smoke tube. This will prevent unnecessary loss of the chemical, reduce the tendency for a solid to form and cake up the tube, and protect the clothing from the corrosive action of the chemical, when the apparatus is being carried in a pocket.

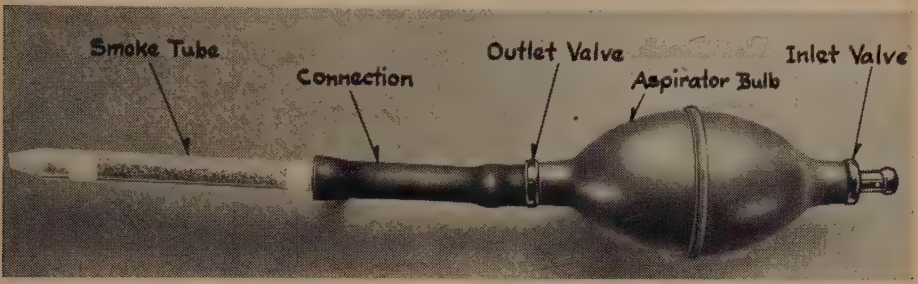

FIG 1—SMOKE APPARATUS. (*Courtesy Mine Safety Appliances Company.*)

Each smoke tube will last for at least a full shift. A partially expended tube may be preserved for later use by detaching it from the rubber hose and bulb, and covering each end with a rubber cap.

MEASURING AIR FLOW

Ordinary vane-type anemometers are suitable for measuring air velocities from 100 to 3000 fpm. A convenient, rapid and satisfactory method for measuring ventilating currents moving at speeds of 100 fpm or less is by use of smoke apparatus. Where the air is flowing in a relatively smooth manner, smoke readings can be made to determine average velocities as low as 10 fpm, but for lower velocities than this, because of diffusion of the smoke, readings are seldom obtainable.

The following procedure is advocated for measuring air velocity with a smoke tube. Select the measuring station along a straight, uniform section of the air course, away from splits and bends. Place vertical chalk lines on the ribs, 15 ft apart, to mark the start and end points for timing the travel of smoke. Then measure the cross-sectional area, as for an anemometer reading.

One man, with smoke apparatus, stations himself along the rib opposite the starting line while the other takes a position downstream, along the rib, opposite the end line. At a given signal, the former quickly depresses the aspirator bulb so as to eject a ball-like cloud of white smoke into the fastest moving part of the air current. The smoke is ejected opposite the starting line and at right angles to the direction of the air current. The end observer notes on the second hand of a watch the time when the smoke cloud is generated, and again when the center portion of the smoke passes the finish line (Fig 2). The average velocity is found by referring to Table 1.

The simplicity of the method is further illustrated in Fig 3, using an air course 8 by 12 ft, having 96 sq ft cross-sectional area. If a smoke cloud travels 15 ft along this air course in 10 sec, the corresponding average velocity (Table 1) is 72 fpm, and the volume 7000 cfm.

The distance of 15 ft for timing the travel of smoke has been found very satisfactory, since it is short enough so that the smoke remains as an integrated cloud. Over much longer distances, the smoke becomes so diffused that the center of the mass cannot readily be distinguished as it passes the end line.

The smoke cloud is injected in the fastest moving part of the air current.

The geometric center of an airway section does not always contain the highest velocity, as the relative temperatures of air and rock, relative smoothness of the different wall surfaces, and other such factors influence velocity. A few trial puffs of smoke in a section, however, will show where the velocity is greatest, and by causing the smoke cloud to travel here, and taking the average velocity throughout the section as 0.8 of the maximum (Table 1) excellent results are obtained.

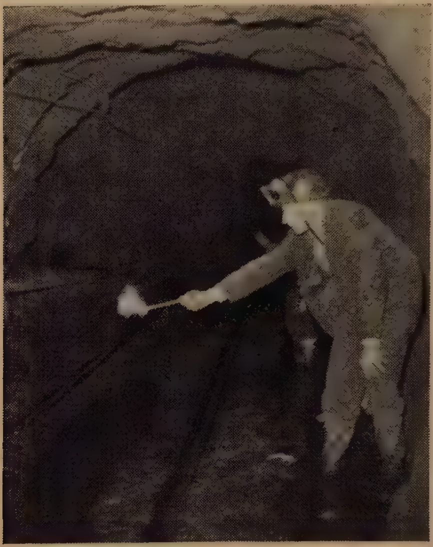

FIG 2—MEASURING AIR FLOW.

Fig 4 shows several typical measuring stations, with the quantities of air passing as separately determined by smoke and anemometer. Only in the portions shown shaded were the air velocities great enough to turn the blades of the anemometer.

The figure, as shown by *a*, indicates that a considerable volume of air—enough to provide the normal requirement for several parties of miners—may be flowing through an air course, and yet cannot be detected with the anemometer.

In contrast with *a*, *b* shows air flowing at the same average overall velocity as in *a*, but, in case *b* the velocity distribution was such that the anemometer turned throughout most of the section and provided a satisfactory air measurement. A similar example is furnished by Fig *e* and *f*. Although the average overall velocity in *e* is less than in *f*, there were practically no so-called "dead areas" for the anemometer in the former, and an accurate measurement could be made, whereas in *f* a true measurement was obtained only with smoke apparatus.

In Fig *c*, *d*, *f*, the anemometer wheel remained stationary in the unshaded portions of the air-course sections, and as would be expected, the traverses have provided measurements that appear to err according to the relative extensiveness of the unshaded to the shaded areas.

In calculating air quantities, some observers use anemometer velocities as recorded on the instrument, without applying the correction from the calibration curve furnished with the instrument.* Quantities which would be derived through using the uncorrected anemometer readings are included in Fig 4 to show the large errors that result when following this procedure in measuring low-velocity air currents.

For the purpose of obtaining up-to-date information for the ventilation maps, ventilation surveys were made recently in local mines. Air quantities were determined at several hundred different places along all main and tributary intake and return air courses. More than 30 pct of all measurements were made with smoke. That the use of smoke apparatus was necessary, in order to obtain so many of the readings, will be better understood when it is realized that standard methods of working local coal deposits require extensive splitting and subsplitting of the main air currents. Resultant velocities in the subsplits normally are not high.

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* If a calibration curve is not available for an anemometer, such a curve can easily be constructed by plotting the anemometer readings versus actual velocities. A sufficient number of actual velocities for establishing the trend of the curve, which will be a straight line, can be determined by walking over a measured distance in still air with the anemometer held at arm's length, and observing the walking time for different speeds, varying from that at which the anemometer will just turn up to the most rapid speed at which a man can walk.

SECONDS FOR SMOKE TO TRAVEL 15' IN ℄ AIRWAY	VELOCITY OF SMOKE FEET PER MINUTE	AVERAGE VELOCITY OF AIR = .8 VELOCITY SMOKE
6	150	120
7	129	103
8	112	90
9	100	80
10	90	72
11	82	65
50	18	14
60	15	12
70	12	10

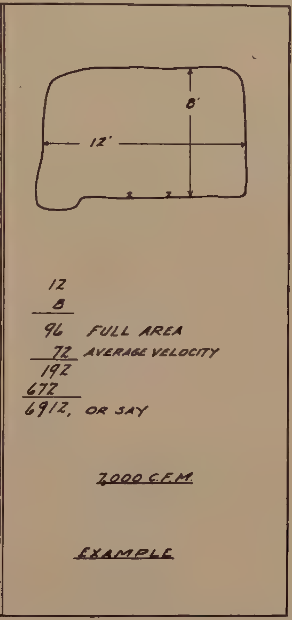

TABLE 1—CONVERSION TABLE. FIG 3—SAMPLE CALCULATION.

No attempt was made on this project to trace the air distribution all the way into each working face. When it is desired to so trace the air, smoke measurements are used almost exclusively for this part of the work, because volumes encountered are relatively small. The common practice is to place each four parties of miners (four faces, with two miners at each face) on a separate split of air.[2]

Anemometer measurements are commonly made in local mines with a 2-min. continuous traverse, and with the instrument held in the hand. Near the middle of the traverse, with the anemometer overhead, the body is pivoted 180°, and a step is then taken backward toward the starting rib, without interrupting the traverse pattern. This permits the operator to cover all parts of the measuring section with the instrument, and the traverse is completed with the anemometer, rather than the body, close along the second rib (Fig 5).

The instrument correction is applied to each reading. For computations, the full area at all measuring stations, except those on a steep incline, is reduced by 4 sq ft to compensate for the space occupied by the man making the traverse.

By exercising care with the method described, results may be obtained that approach those of the multisectional method of measuring air flow.

When making an anemometer measurement in a steeply inclined opening, a man has difficulty in maintaining his balance and footing. Frequently the procedure he uses in measuring is governed more by what body movements he can

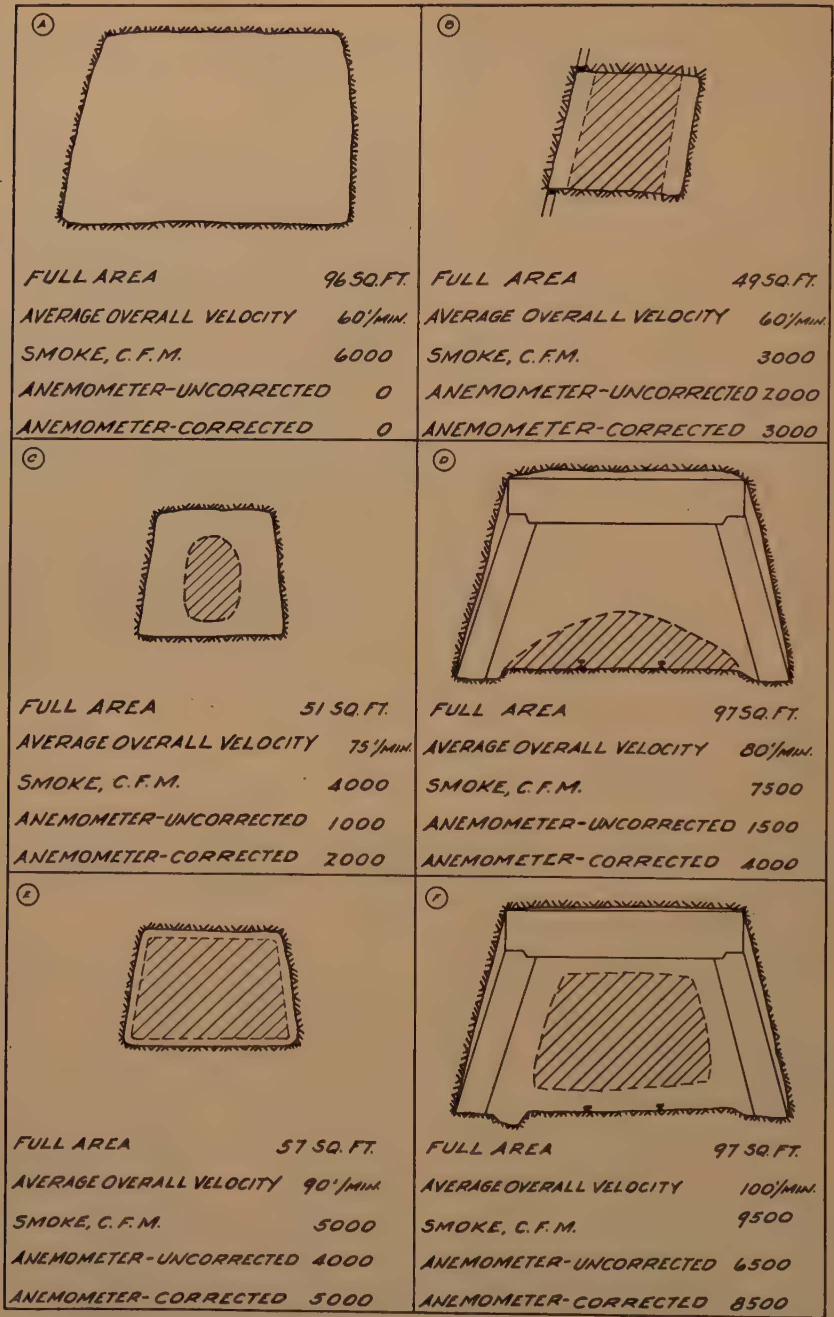
Ⓐ
FULL AREA 96 SQ. FT.
AVERAGE OVERALL VELOCITY 60'/MIN.
SMOKE, C.F.M. 6000
ANEMOMETER-UNCORRECTED 0
ANEMOMETER-CORRECTED 0
Ⓑ
FULL AREA 49 SQ. FT.
AVERAGE OVERALL VELOCITY 60'/MIN.
SMOKE, C.F.M. 3000
ANEMOMETER-UNCORRECTED 2000
ANEMOMETER-CORRECTED 3000
Ⓒ
FULL AREA 51 SQ. FT.
AVERAGE OVERALL VELOCITY 75'/MIN.
SMOKE, C.F.M. 4000
ANEMOMETER-UNCORRECTED 1000
ANEMOMETER-CORRECTED 2000
Ⓓ
FULL AREA 97 SQ. FT.
AVERAGE OVERALL VELOCITY 80'/MIN.
SMOKE, C.F.M. 7500
ANEMOMETER-UNCORRECTED 1500
ANEMOMETER-CORRECTED 4000
Ⓔ
FULL AREA 57 SQ. FT.
AVERAGE OVERALL VELOCITY 90'/MIN.
SMOKE, C.F.M. 5000
ANEMOMETER-UNCORRECTED 4000
ANEMOMETER-CORRECTED 5000
Ⓕ
FULL AREA 97 SQ. FT.
AVERAGE OVERALL VELOCITY 100'/MIN.
SMOKE, C.F.M. 9500
ANEMOMETER-UNCORRECTED 6500
ANEMOMETER-CORRECTED 8500

Fig 4—Typical stations where low-velocity air currents were measured.

Both the smoke method and the anemometer were used. Only in the shaded portions were velocities great enough to turn the anemometer wheel.

make without falling, rather than what he would desire to do from a technical standpoint. He must use his judgment, in each case, in deciding how much should be deducted from the full cross-sectional area of the measuring station, for the space occupied by his body.

When the required data can be obtained elsewhere, it appears best to avoid making anemometer measurements in steeply inclined openings. The flow through many inclined openings can be suitably ascertained by measuring the air flowing in the level air course, on each side of the chute or raise connection, and using the difference in quantities determined, as the amount flowing through the inclined opening.

When a reading is to be obtained in a regulator opening, a 1-min. traverse is used. The corrected velocity is then multiplied by an area, which may be 0.5 sq ft less than that of the opening, the reduction here compensating for space taken up by the operator's arm. A second method, which produces like results, is that of holding the anemometer in the center of the regulator opening, and computing the average overall velocity at 85 pct of the corrected center-point reading.

Quantitative determinations by both smoke and anemometer have consistently provided results that balance with a deviation of but 5 pct. As an example of what is meant here, let us assume a condition of split flow with air currents 1 and 2 joining to form current 3. The quantity of air in current 3, as measured by an anemometer, can be expected to equal, within 5 pct, the sum of currents 1 and 2, as independently and appropriately determined, regardless of whether both of these were made with smoke, both with an anemometer, or one by smoke and the other by anemometer.

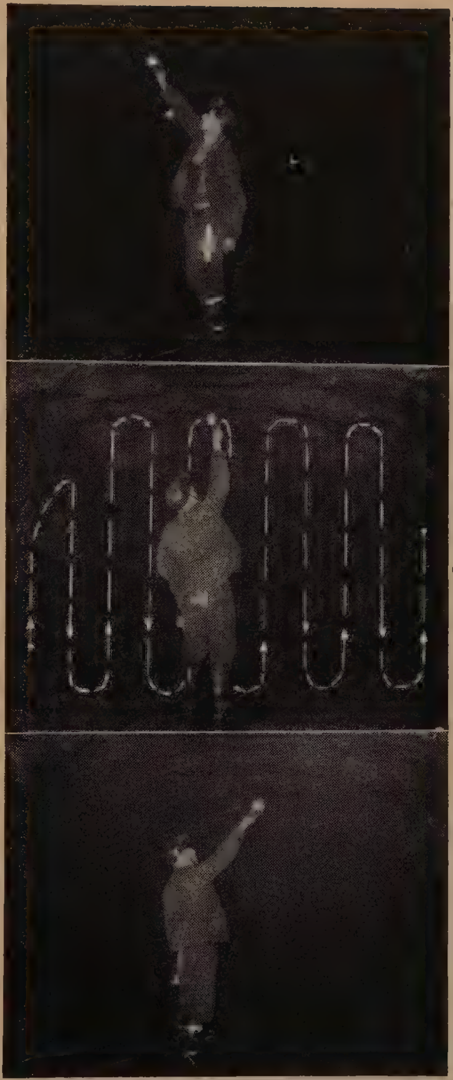

FIG 5—POSITIONS OF OPERATOR IN USING ANEMOMETER.

Near the middle of the traverse, with the anemometer overhead, the body is pivoted 180° and a step is taken backward toward the starting rib without interrupting the traverse pattern. This permits the operator to cover all parts of the measuring station, and the traverse is completed with the anemometer rather than the body close along the second rib.

Quantities are always permanently recorded in numbers rounded to the nearest 500 cfm for values under 20,000 and to the nearest 1000 cfm for values over 20,000, because greater accuracy than this is not justified by the attainable accuracy of the measured data (height, width, and velocity).

Leakage Detection

What percentage of the air that passes through the main fan should reach the working faces? In view of the differences in the character of mines, a standard indicating poor performance at one, might reflect reasonably good performance at another. Although no definite percentage is known to be established for mines in general, data have been released in the past showing that where ventilation practices were lax an almost unbelievingly small percentage of the air handled by the fan reached the working faces. Frequent mention is made in the technical literature of the inefficiency and high cost of air being short-circuited, and it is well for us to keep this matter in mind, because only with much patience and care is it possible to conduct air through the mines, and to the faces, without losing much of it along the way.

Fig 6—Making a test for air leakage at a door.

An outstanding application of ventilation smoke apparatus is in locating leakages in the mine air system. To test for air-tightness of a brattice, for example, it is necessary only to spray a quantity of smoke on the high-pressure side of the structure, and observe any places where the smoke draws through. Testing with smoke at openings into the gob, at doors, traps, sheets, or along line brattices, provides the observer with an unmistakable answer as to where, and to what extent, air is being short-circuited (Fig 6, 7).

Several years ago, a ventilation survey in one of the local mines disclosed a much greater quantity of air to be passing through a main exhaust fan than could be measured as a total return inside the mine. Anemometer readings were checked and rechecked, tests were made of brattices along the airhole, and the airhole and fan were inspected. The difference in quantities was verified, but there was no indication of where the trouble existed.

The overlying surface was then carefully inspected, and here, with the aid of smoke apparatus, air leakage was found. In various spots over almost an acre of ground, smoke was observed seeping between rocks and through small cracks in the surface. A large volume of air was filtering through the broken ground and finding its way into the airhole, around the lagging over several sets of timber.

This leakage, once discovered, was quickly eliminated by placing a layer of clay, several feet thick, over the broken ground. The cost of the job was paid for by electric power saved in driving the fan during the following five months of operation.

Observing Air-flow Characteristics

In local mines there are many openings interconnecting the various levels and the surface, and through these openings there will customarily be a flow of air. This condition arises, of course, because air, like water, distributes its flow among any number of available channels, in an inverse proportion to the relative resistance of each, and in addition, natural draft pressures may assist, balance, or overbalance the mechanically generated flows in certain parts of the system. The flow may thus be affected by changes in temperature and barometer as well as a change of conditions inside the mine. Such openings may then be either of benefit or harm, depending upon the amount and direction of flow. Smoke apparatus has been of much value in analyzing these conditions, and has permitted the making of sound decisions as to whether the openings in question should be bratticed, regulated, or kept open.

When fighting mine fires, rigid control of slow-moving air currents is a necessity. Here again, the use of smoke has provided a practical method for easily and quickly determining direction of flow, tracing air currents, and estimating air quantities.

Finally, convection currents, turbulence, velocity distribution in airway sections, and other such conditions are revealed to an observer using the apparatus as they would probably be in no other way.

Summary

Through ventilation engineering work, conducted in large anthracite mines operated by the Lehigh Navigation Coal Company, Inc., it has been found that:

1. The ordinary commercially available anemometer cannot be relied upon to provide an accurate measurement of a low-velocity air current. As much as 10,000 cfm of air, flowing in a standard-size 8- by 12-ft air course, may be inaccurately determined through an anemometer reading, and as much as 6000 cfm flowing through the air course may cause no movement whatsoever of the anemometer wheel.

2. Peculiar local conditions for mining have necessitated extensive subsplitting of the ventilating currents, and the attendant low velocities are found throughout considerable portions of the mines. Consequently, when air quantity measurements have been secured for analysis of the ventilation systems, the smoke tube has been used because with the anemometer alone more than 30 pct of the total number of measurements either could not have been made at all, or would have been inaccurately determined. These measurements were satisfactorily obtained by using smoke apparatus.

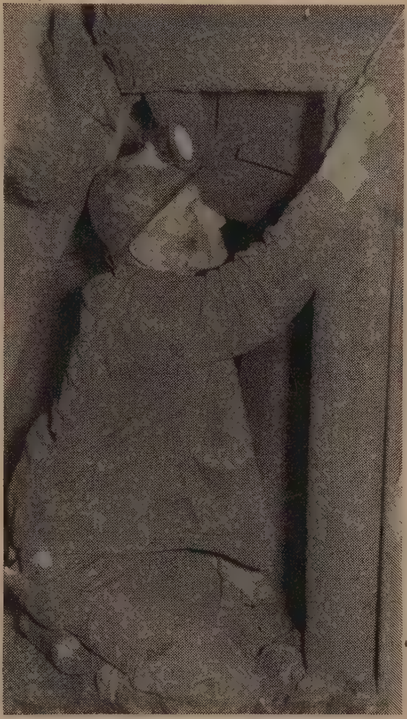

FIG 7—TESTING FOR AIR LEAKAGE ALONG THE PLANK PARTITION IN A CHUTE.

When a chute is being driven, air is coursed up through the manway compartment to the face, and returns down the coal chute compartment.

3. The smoke-tube apparatus provides a ready means for locating leakages in the ventilating system; its application in this connection is of extremely high value.

4. Many air-flow characteristics can actually be observed by injecting smoke into the air.

Smoke apparatus has a definite place in practical mine ventilation work; it should be placed on the "must" list for the operating and engineering personnel who are responsible for maintaining good mine ventilation.

ACKNOWLEDGMENTS

The writer acknowledges, with thanks, aid given in the preparation of this paper by Mr. D. C. Helms, Production Manager, Mr. Norman Richards, Coaldale District Superintendent, and Mr. Thomas J. West, Tamaqua District Superintendent, all of the Lehigh Navigation Coal Company Inc., and to Prof. William B. Plank, Head of Department of Mining and Metallurgical Engineering, Lafayette College.

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1. S. H. Katz and J. J. Bloomfield: Sulphur Trioxide Smoke Tubes for Determining Air Currents. U. S. Bur. Mines *R.I.* 2505 (1923).
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DISCUSSION

(*C. Smith presiding*)

C. SMITH—Mr. Beckwith, that is an excellent paper and is much broader in scope than I thought it would be.

G. E. MCELROY*—I am particularly grateful to Mr. Beckwith for presenting this paper on the use of smoke tubes in ventilation surveys, as I have been a strong advocate of their use since they were developed at my request in the spring of 1922. Before that time I had several years experience with less-convenient methods of generating smoke clouds for tracing and measuring low-velocity air currents in metal mines, where about 80 pct of the air-distribution detail ordinarily must be determined by some such means. Since that time I have used smoke tubes to great advantage in ventilation surveys of metal mines and on inspection trips through coal mines, including a two-day trip to the LNC Co. mines in June 1934. My methods of smoke measurement differ only in unimportant details from Mr. Beckwith's in that I prefer a distance of 25 ft, generate small clouds at the quarter points of the cross section, and use a factor of 0.9 for this position.

The method of anemometer traversing recommended by Mr. Beckwith is very good as the anemometer is always at arm's length from the body and never in line with it. In correcting readings, however, 5 to 10 pct, depending on the size of the airway, should be deducted from the reading to allow for higher velocities caused by the hand holding the instrument, but *no* deduction in area of cross section should be made for the body of the observer because it affects the velocity distribution only upstream and downstream and immediately adjacent to it. There is no noticeable effect at the anemometer position at arm's length from the body.

It is hoped that the publicity given by this paper to low-velocity measurements by smoke-cloud methods will make both mine officials and mine inspectors more conscious of the value and utility of these methods.

A. T. BECKWITH (author's reply)—Mr. McElroy's valued comments are much appreciated.

In connection with applying a correction to an anemometer traverse, Mr. McElroy explains that no deduction in area of cross section should be made for the body of the observer, but rather, a 5 to 10 pct deduction, depending mainly on the size of the airway, should be made to correct the reading for the effect of hand holding the instrument.

In U.S.B.M. *Bulletin* 385—Engineering Factors in the Ventilation of Metal Mines—Mr. McElroy mentions that with anemometer traverses (comparable with these under discussion) he has satisfactorily determined air quantities by applying a correction factor to reduce the results by 10 pct for openings of about 40 sq ft cross section, while for larger openings the relative effect on "method factor" is less. Correspondingly, he recommends the use of smaller percentage deductions for correcting measurements in larger openings.

The standard deduction of 4 sq ft from the measured cross sectional area, which I use, is mathematically equal to a 10 pct reduction when applied to a 40 sq ft opening, and to a 5 pct reduction for an opening of 80 sq ft.

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* Supervising Engineer, Bureau of Mines, Pittsburgh, Pennsylvania.

Mr. McElroy is correct, however, in pointing out the fact that the effect of the observer's body in the traverse plane should not be considered as a pure reduction in the area of cross section.

A. L. Barrett*—I was wondering how Mr. Beckwith arrived at the figure of 0.8 pct times maximum velocity. After looking at some of these pictures, I should like to know the varying behavior of this factor in certain average airwaves.

A. T. Beckwith—Different investigators have pointed out that for conditions of uniform flow, with maximum velocity at the center, the average overall velocity in an air passageway will equal about 80 pct of the maximum velocity. By actual use of the figure 0.8 times maximum velocity, with many measurements and under conditions shown by the drawings to which Mr. Barrett has made reference, I have always obtained results which have balanced out, and have thus accepted this figure as being correct.

H. P. Greenwald†—That figure has much experimental work behind it. E. Ower gives a discussion for smooth pipes on pages 42–44 of his book "Measurement of Air Flow." G. E. McElroy of the Bureau of Mines also has considerable data. The factor is affected by changes in area of the passageway. At a constriction, flow increases faster close to the walls than at the center and the factor rises locally until the distribution of air is readjusted further downstream. The reverse is true on expansions.

I ask the author (and I may duplicate other questioners here because I was not in the room during the presentation of the whole paper) whether he set up air measuring stations at principal locations where measurements would be repeated from time to time, whether he kept these same stations all the time, and whether initially he examined the area in the general vicinity of such stations in a search for the best possible location. What prompts these questions is that in one mine we tried to get an air measurement at one spot and found the air crowded entirely to one side of the entry; there was an eddy current running in reverse direction at the opposite rib. We examined conditions upstream and found a huge bump on the rib left by irregular mining. This deflected the air current to the opposite rib and there was an eddy current in the lee of that bump.

I think that to obtain best results it would be wise to have established measuring stations at the more important points, the characteristics of which have been investigated. An anemometer reading can then be taken at a single and definite spot in the cross section and the average velocity obtained by use of a multiplier determined for that station.

A. T. Beckwith—When possible, that is done. I might add that one of the things we usually go into, before setting up a station, is to take a look in both directions along the air course over the available ground, and then select the most appropriate place. The smoke apparatus may then be used to determine whether there are any unusual conditions that obtain there before going ahead with the anemometer.

C. Smith—I suppose one thing that might be proposed in connection with your low velocity measurements would be to use a slow speed anemometer. The one you have in mind is a standard 4-in. one. A person can get a 3-in. anemometer but he would carry two pieces of equipment and would not have the advantage of smoke procedure.

J. J. Rutledge*—I just want to ask where this apparatus can be purchased.

A. T. Beckwith—The Mine Safety Appliances Co., Pittsburgh, sells the apparatus. There are probably other firms who also sell it.

C. Smith—Mr. Beckwith you are not a member of our ventilation committee but if you will accept, we will put you on it. We have drafted Mr. Sturges and we are going to draft you, too.

The stimulus of this paper makes me think we should study anemometry. There are a lot of tasks and problems that should have been answered a long time ago. It is a basic thing that we slough off and, as you know, there is some terrible work being done in measuring and reporting rates of flow underground. I think that may be the gist of our next year's study.

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* Pittsburgh Coal Co., Library, Pa.
† U. S. Bureau of Mines, Pittsburgh, Pa.

* Maryland Bureau of Mines, Annapolis, Maryland.

Core Drilled Shafts for Ventilation and Emergency Escapeways

By F. C. Sturges,* Member AIME

(New York Meeting, March 1947)

For some time mining engineers have been interested in the possibility of using small diameter shafts, sunk by core drilling, as aids to ventilation and as emergency escapeways. The possibilities are interesting to contemplate, but new methods involve unknowns, and interest tends to remain passive until these unknowns are solved. Recently we have completed two 36-in. diam core drilled shafts, and wish to make the data on these available to the industry.

Both shafts were drilled for the McLain Fire Brick Co., one at its Buckeye mine and the other at its Colonial mine. Both mines are in the fireclay underlying the Lower Kittanning coal. Both mines have been in operation for a number of years and have required expensive maintenance work on return airways. It was decided that considerable savings could be effected by short-circuiting the air to the surface through small shafts. It was also realized that these holes, located near the present workings, would be of added value as emergency escapeways.

Table 1—*Geologic Horizons Penetrated*

Conemaugh Formation	67 feet	Mahoning Sandstone
Allegheny Formation	173 feet	Upper Freeport Coal Butler Sandstone Bolivar Clay Lower Freeport Coal Freeport Sandstone Upper Kittaning Coal Worthington Sandstone Middle Kittaning Coal Lower Kittaning Coal L. Kittaning Underclay

Originally, 48-in. diam shafts were considered, but actual computations showed 36-in. diam shafts were more than ample at both mines.

The shaft at the Buckeye mine is 240 ft deep, and that at the Colonial mine 258 ft deep. Both holes were started in the lower part of the Conemaugh formation and penetrated Conemaugh and Allegheny strata.

The holes were started through the unconsolidated materials overlying bedrock by constructing a concrete collar with a wall thickness of 8 in. Then the shafts were drilled by the shot-core drill method. An annular groove was cut in the rock by the abrading action of chilled steel shot run under the shoe of the core barrel. The drilling was done in stages, the core barrel drilling the groove for a certain distance depending on conditions encountered, and then the cores broken off and taken out in sections. The core barrel had a 36-in. o.d. and the finished diameter of the shaft measured almost 37 in.

The major questions that could not be answered until the completion of the first shaft were, (1) how much ground water would be encountered, how would it hinder the drilling, and how difficult would it be to seal off, and, (2) how would the walls of the hole stand up. The holes of course could be cased or lined, but that is considerable added expense in any shaft.

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[*] Vice President, Pennsylvania Drilling Co., Pittsburgh, Pa.

Ground water was encountered in quantity in crevices in the Lower Mahoning sandstone. As a precaution, a 3-in. hole 50 ft deep was drilled and pregrouted before starting the drilling of the first shaft. Just how much water the pregrouting accounted for is unknown, but when the shaft reached final depth there was an inflow of about 35 gpm, almost all of it coming from the pregrouted section. Experimenting with a new method of grouting, all visible points of water entry were grouted off at pressures up to 250 psi. After grouting every point where water could be seen coming in, the walls were still wet from the accumulation of minute amounts coming through pores and hair cracks too small to be detected, but the total water accumulating amounted only to about ten drops per minute falling into the mine below.

The second shaft, at the Colonial Mine, still had an inflow of 5 gal per hr when all points of visible inflow had been grouted off. Measurements were made of the total accumulation at different points in the shaft, and it was found that 80 pct of this was coming from a 12-ft section at a depth of 70 ft. This section was lined with a thin steel shell 35¾ in. i.d. and grout behind it. When this was done, the total seepage from the shaft measured only 1 gal per hr.

These shafts are to be used to exhaust the air from the mines, so the drying up of the holes was more perfect than necessary, but it was thought desirable to demonstrate that the holes could be dried up to be suitable for air intake.

As to caving, the solid rock wall of the holes indicates its strength. Its circular shape gives perfect arch action, and it is smooth cut instead of showing the shattered appearance that usually results from blasting. Before anything was done to the walls, there was no danger involved in being lowered down the open hole for over 200 ft. Near the bottom, just under the Middle Kittanning coal, there was a

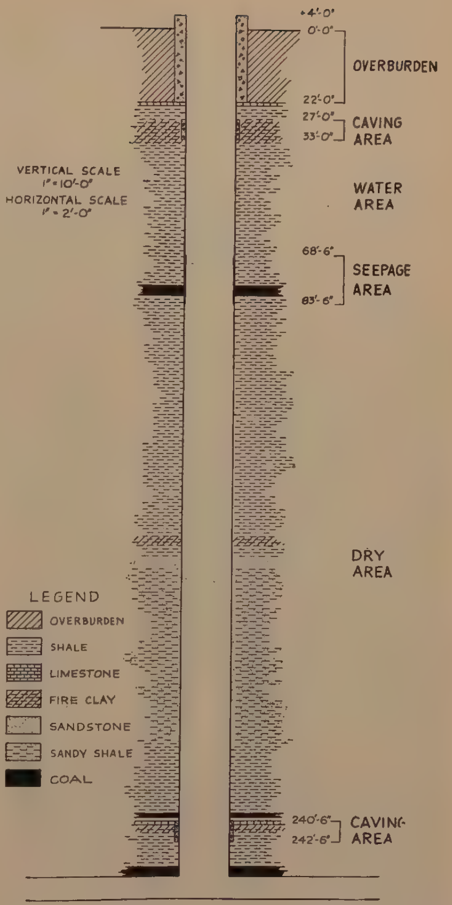

Fig 1—Sectional view 36-inch drilled shaft. Colonial mine McLain fire brick company.

Overburden: Concrete collar 8-in. wall thickness

Caving Area: Reamed weathered shale to 42 in. diam and lined with cement placed under pressure behind removable liner. Finished wall is flush with original hole.

Water Area: All points of visible inflow grouted off under pressures up to 250 psi. Balance from seepage 1 gal per hr which was ignored.

Seepage Area: All visible inflow grouted off. Balance of inflow 4 gal per hr, sealed off with 36 in. o.d. by 35¾ i.d. Steel shell pressure grouted in permanently.

Dry Area: This section was dry except at a few points which were pressure grouted.

Caving Area: Reamed to 42 in. diam and lined with cement.

Drainage into mine—1 gal per hr.

stratum that disintegrated rapidly. Dr. J. M. Schopf of the U. S. Bureau of Mines, who logged both holes, classified this caving bed as "clay, light gray, very soft, much slickensided, with fist-size lime nodules." This section was reamed and lined with 3 in. of cement.

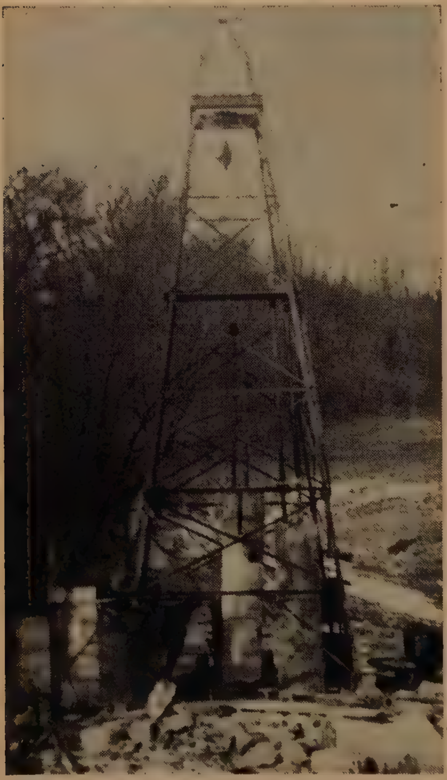

Fig 2—Shaft drill rig with 10 ft 6 in. length of core hanging in derrick.

The shale and fireclay immediately under the concrete collars were badly weathered. This section also was reamed and lined. Since this top weathered zone can not be considered in normal condition, and the concrete collar might have been extended through it, evidently there was only one stratum encountered in each shaft that required support. This was accomplished by reaming and lining a section about 7 ft long in each shaft. This is of particular importance, because approximately 80 pct of the wall surface in the Colonial shaft was coal, fireclay, clay shale and carbonaceous shale.

Fig 3—View of derrick floor.

While work was in progress on the first hole, there was a possibility that some shale beds encountered might need some protection against eventual weathering. Study was made of different protective coatings that might be applied. Gunite, cement paints, other paints and plastics were considered. No bed was encountered that appeared to need this protection, but in anticipation of future requirements on other holes, some experimenting was done. Of particular interest was a paint that could be used on the damp walls. This was brushed on as a test and offers promise of valuable use in future holes with questionable shale beds. A circular hole could be painted every two or three years with a rotating spray gun at very little cost.

The cost of these shafts was dependent on conditions encountered, depth of concrete collar, amount of grouting, reaming and lining required, and others. The final figures showed a cost of approximately $10,000 for each shaft.

The McLain Fire Brick Co. has ordered a fan for installation near the foot of the Buckeye hole but the installation is not yet complete. Therefore we have no actual measurement of the coefficient of friction in the hole. It would appear that the coefficient will approximate that for smooth concrete, about 0.25×10^8. Using this figure in our formula, the shaft at the Buckeye mine could then be expected to pass 20,000 cfm of air at 1 in. water gauge, 34,000 cfm at 3 in. water gauge, 48,000 cfm at 6 in. water gauge, and so on. These figures on air passage would be approximately doubled for a 48-in. hole. A measured coefficient should be available shortly.

It is planned to install emergency hoisting equipment at these shafts so that the men may be taken out of the mine quickly in case of need. These two holes were considered worthwhile from the standpoint of ventilation alone, and their additional worth as escapeways is difficult to evaluate. In any given case, the value would be dependent on conditions in the particular mine, distance of the workings from other exits, and similar factors. If disasters could be foreseen it would be of infinite value to have at least one emergency escapeway in every mine slated for a fire or an explosion.

It was demonstrated to visitors that four men of average size could stand side by side in a cage built for one of these 36-in. holes. A two or three deck cage would be feasible.

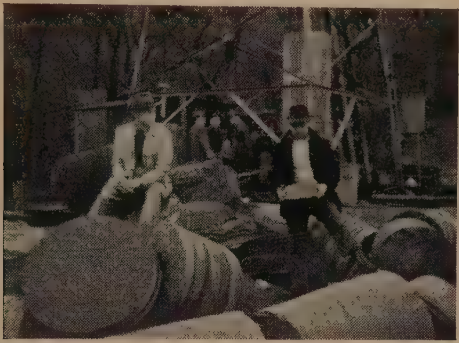

FIG 4—VISITING GEOLOGISTS ON THE CORE PILE. CORE MEASURES 32 IN. IN DIAM.

These two shafts were drilled with 36-in. diam tools because actual computation showed that they were of ample size for the amount of air to be passed through them. It is believed that at many other mines 48-in. diam holes will be preferable, as they will pass approximately double the amount of air and yet the drilling cost is not much greater.

The conditions encountered in these two holes might be either better or worse than those to be encountered in future core-drilled shafts, but actual experience gained from these two holes can be summarized as follows:

1. Ground water, which is generally encountered in this area, is not a serious obstacle to the drilling of the holes by the methods employed.

2. Ground water can be grouted off after the hole is drilled much more effectively and at a very small fraction of the cost of pregrouting.

3. The circular rock wall, smooth cut and unshattered, gives a perfect arch

action, and only an occasional stratum will require artificial wall support.

4. Artificial support of a weak stratum can be obtained economically by reaming and lining the individual bed.

5. Any questionable stratum that might weather in time can be kept painted, a minor undertaking in a circular hole with smooth walls.

6. The fact that four men of average size can stand side by side in a cage built for a 36-in. hole demonstrates the value of the holes as emergency escapeways.

A final conclusion is that present developments of methods of drilling these holes forecast decreasing costs in contrast to the general trend.

Thanks are due the McLain Fire Brick Co. and its Chief Engineer, Mr. J. B. Blewett, for permission to make this information available to the industry.

DISCUSSION

(*C. Smith presiding*)

C. Smith—Gentlemen, I am sure there are a number of questions in your minds as a result of listening to this excellent paper. I am sure Mr. Sturges will be glad to answer any questions.

L. E. Young*—We are indebted to the Committee for bringing before us this very interesting subject. Recently, I had the pleasure of going down one of these 36-in. shafts, and while the walls were smooth and sound, I must confess that for a brief time I experienced claustrophobia. At a gassy coal mine near Rosita, Mexico, it is the practice, I understand, to put down deep boreholes to tap the remote mine workings and exhaust the mine gases through these holes instead of returning the mine air to the conventional air-shafts. Possibly some one in attendance can give us more information about the drilling of the holes at Rosita and the ventilation practice followed.

J. J. Rutledge†—I presume that is the property over which Mr. Hughes has charge—the lignite mine?

L. E. Young—No. This is a bituminous coal property under the management of a member of the Institute, H. D. Bubb, Jr., El Paso, Texas, who is Manager of the Coal and Coke Department of the American Smelting and Refining Co. I understand this company is doing a splendid job in ventilating a gassy mine by using multiple boreholes to exhaust mine air.

J. J. Rutledge—One of the most gaseous mines in a pitching seam in eastern Oklahoma was relieved of methane in advanced workings by drilling 6-in. diam churn-drill holes from the surface down to the Lower Hartshorne coal seam. These holes were drilled so as to strike the coal in advance of the heading a very short distance; in some cases the drill-holes only extended to a 1-ft thick seam of coal which was about 6 to 8 ft above the main seam of the Lower Hartshorne. The Degnan-McConnell Coal Co. owned the churn-drill and I was advised by Mr. McConnell that the cost of drilling the holes was about $1.00 per linear ft in depth.

This method seemed to free the mines of methane in advance workings. Most of the shafts and the slope openings were on the intake and when it was suggested to some of the operators that they follow the example of the Degnan-McConnell Co. and drill bore-holes from the surface to free their mine workings of methane, they objected saying that such bore-holes would not relieve the mine workings and that the outside air would enter the mouth of the drill-hole at the surface and thus prevent the circulation of air.

I saw the 30-in. drill-hole drilled by the Pennsylvania Drilling Co. on the Confluence Dam at Confluence, Pa. There was one hole through the Morgantown sandstone and through the kindness of the engineer in charge, the Bureau of Mines of Maryland was given a 1-ft thick cross section of this coal.

I think this is one of the finest developments in mining engineering in recent years. However, 6 or 8 or 10-in. diam holes will relieve gaseous areas if you cannot afford these larger diameter holes.

Some will say that the outside air will come down the bore-hole and prevent the exit of the gas from the mine workings but this has not been the writer's experience.

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† Maryland Bureau of Mines, Annapolis, Md.

In western Maryland we have a long fire clay tunnel driven directly into the mountain for a distance of 1000 or more feet and in warm weather it would not be possible to operate this tunnel if it were not for a 10-in. borehole drilled from the surface and giving exit to the air in the workings.

It has long been the practice in the Joplin, Mo., area to drill 17-in. churn-drill holes from the surface to the mine workings for ventilating these lead and zinc mines.

C. A. FREDELL*—In answer to Dr. Young's question, I cannot give first-hand information, but we have a man in our organization who worked in Mexico in that mine and started the drilling.

Originally they put down a 12-in. hole and used that as a pilot hole and successfully reamed it to 24, 42, and up to 60 in. A number of holes were put down that way and with much success and speed. Their present practice is to start with an 8-in. hole, ream to 24 in. and enlarge by drilling and blasting, dropping the muck down through the 24-in. hole. They are getting faster speeds that way.

C. SMITH—In that way, they are losing the smooth periphery that Mr. Sturges likes.

C. A. FREDELL—That is right.

C. SMITH—Did you give us the rate at which you put down this hole, Mr. Sturges?

F. STURGES (author's reply)—No, I did not.

R. MORRIS†—How did you keep the hole straight?

F. STURGES—We did not make any particular effort to keep it straight other than to start the core barrel straight.

L. E. YOUNG—It looked fairly straight and smooth. From the bottom of the 250-ft shaft, looking up, there was no evident curve in the hole.

A. L. TOENGES‡—I was in Mexico in January. The present practice is first to sink through the overburden and then drill an 8-in. hole to the mine openings. The hole is reamed or countersunk, as they call it, to 24-in. with a Christmas tree arrangement equipped with Hughes bits. Then they blast, shooting the muck down to cars in the mine. If the drilling is very good, they will ream again or countersink to 42 in. and then blast around, making a shaft 9 ft in diam. The shaft is then lined with concrete to an inside diam of 7 ft. They use four men on the shaft sinking. Their rate of progress on a 24-in. hole is about 1 ft per hr and on a 42 in. hole, it is about 6 in. per hr.

L. E. YOUNG—Will you kindly mention the ventilation aspect of this in regard to the gases?

A. L. TOENGES—I cannot give too much information. In fact, I cannot give any information, for various reasons, on their degasification program. However, I will say they sink an airshaft for about every 180 acres. At the present time they have eleven shafts, five intake and six return. There is no electricity used at the faces in the mine.

C. SMITH—Are they using these shafts with escape equipment—ladders, stairs, or hoists?

A. L. TOENGES—They are planning on a small hoist as an auxiliary.

M. D. COOPER*—Mr. Sturges has given a modest report on a splendid job. Anybody would be well rewarded by visiting these shafts. They are not hard to reach from Pittsburgh. He is a psychologist and refers to them as shafts rather than boreholes because boreholes give us a squeamish feeling, thinking about going into the cage. However, in starting down, and especially when coming up, the safety factors that are involved are impressive. It is an excellent job. The most outstanding feature about it is the smooth surface. Another is the reasonably plumb condition. It is not actually plumb, but it is near enough to be considered so.

There are one or two items about which Mr. Sturges may give us a little more information. What is the maximum diameter that could be drilled? Does he care to comment on the simple approximate cost estimate that he gave me at the time I visited the shaft? At that time he said it would be possible to consider the cost at $1.00 per inch of diameter per foot of depth. I would like to have a word or two from him on both of those subjects.

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† Gauley Mt. Coal Co., Ansted, W. Va.

‡ Fuels and Explosives Service, U. S. Bureau of Mines, Pittsburgh, Pa.

* National Coal Assn., Pittsburgh, Pa.

F. STURGES—In connection with the cost, there are many variables which enter into it. I have been giving people a figure of $1.00 per inch of diameter per foot of depth as a rule-of-thumb yardstick. An actual cost estimate for conditions to be encountered on an actual job might be considerably higher or lower.

As to the other question, we made a proposal at one time to drill some 6-ft diam shafts. The largest we have drilled is 36 in. in diam. but 48 in. is what we are now prepared to drill. I think we will get up to 6 ft and maybe we will go further, but, first of all, we would like to drill the 48-in. diam and then go, perhaps, to 60 in., and in that fashion, step it up.

In connection with the shaft work that Dr. Rutledge mentioned, there is an entirely different scheme. A man from California has drilled 5½ ft diam shafts to depths of over 2000 ft. He has a machine that goes down in the hole. For that reason, he is up against a water problem.

We must look at this thing from the standpoint of what we would be up against in our own region and I do not think it would be possible to start one of these holes and not expect to hit water. Therefore, we think it should be a surface drilling operation.

J. RUTLEDGE—Could you go into details on that?

F. STURGES—The California man to whom I referred has his core barrel attached to an operator's cage which also goes down in the hole. In the operator's cage is an electric driving motor. The entire drill is lowered into the hole on a cable, the electric line being run down for operation of the motor. He jacks the cage against the wall of the hole and starts his tools operating. Everything is done down in the hole. If he should hit an unexpected amount of water, I do not know what he would do.

If that type of drill were used in the Pennsylvania coal fields, I think the site would have to be pregrouted. You know that is an expensive proposition; that is why we are approaching it from the standpoint of leaving the equipment on the surface and not worrying about the water because we can go down, after the hole is completed, and shut that water off very quickly and economically.

If you have a crevice with an opening of one inch, a lot of water can come in suddenly. The usual condition of these crevices has been seen on the sides of these shafts. Horizontal crevices go completely around the hole, but they are sealed for most of the circumference. This condition has been noted by visitors to these holes. There may be one or two places where water will be coming in in jets, but we can close those jets by using a very small amount of cement just exactly where it is needed.

C. SMITH—We have a 15-ft diam hole. In this particular case we have a center core and four boreholes around that. We work on angle repose which is ordinary oil country operation and a very characteristic method, while using a fast setting cement. In that way, we get a complete shut-off.

By angle repose, I mean putting a wall in there and pumping against that wall until you get a complete shut-off with a 1000 lb pressure.

In order to demonstrate the feasibility of that particular thing in experimental work (and I am anticipating a paper that will come out), we went in and did a grouting. Those cores are probably as illuminating to look at as you will ever find. They show not one crevice but a multiplicity not only of volumetric control but of capillary control in a thoroughly saturated 500 ft of column with water.

As you know, that is one of the very wet areas. At no point, except where we had one break-through caused by bad technique and bad cement, did the calculated amount of cement put in go beyond this 40° angle repose.

Therefore, I think the techniques which are now coming into the industry and which are carried over from experimental work in the oil industry answer the question of pregrouting the shaft successfully and economically, and that can be done with possibly 8 or 10 pct of the cement used in a grouting job.

F. STURGES—There is another part to the problem. There is no question about limiting the amount of grout put in. If a hole is put down here and another one over here (*indicat-*

ing drawing on blackboard), on the circumference of that 36-in. shaft, it is very questionable whether this crevice would ever be shut off, since the odds are that the grout holes will not encounter any open part of the crevice.

C. Smith—I do not want to argue the question, but here is a case in point. This was grouted under pressures up to 2000 lb, starting with a low pressure of 200 lb. We were doing each horizon or each strata as we came to it. We would measure the amounts of material going in and then, in order to check this, we would drill subsequent boreholes.

In these, you find certain chemical phenomena. You find where your cement has penetrated; you find in one borehole that you have not achieved a complete shut-off; you have one thirsty formation that has taken a little more than you needed. Again you grout that particular area or horizon. You go ahead and drill this one (*indicating drawing on blackboard*) and by the time you get over to this (*indicating drawing on blackboard*) you find you have practically a complete shut-off, and if you do not, you grout this (*indicating drawing on blackboard*). These, then, act not only as checks but as an additional measure for completing the job started here. On the basis of overall costs of shaft sinking this would seem to have a preferred position. Does that answer your question? The approach is towards mathematical infinity. As a matter of fact, I would say that three holes normally would take care of a shaft.

F. Sturges—Does anybody have experience in shaft work in the East?

C. Enzian*—I should like to say in support of Mr. Sturges' statement that when you start grouting water-bearing strata, you do not know where you will end. I think Mr. Sturges has in mind the strata that occurs in the central part of Pennsylvania. I have had experience in trying to grout off a shaft, 16 by 32, that had struck a water-bearing strata. In fact, it was leached-out limestone strata and, of course, we proceeded with the best equipment that a very responsible rock contractor from Pittsburgh had devised and we pumped grout into the water fissures up to 400 lb pressure.

We thought we were doing a pretty good job,—and we were; a few days before we had finished, the farmer from the adjoining farm came down and told us we would have to stop as we had ruined his spring nearly ⅓ mi away.

Regarding the question of boreholes, I wish to say that the drilling of boreholes for the elimination of accumulated gases, whether explosive gases or not, especially in the steep pitch mining area of the anthracite region, has been done for nearly 50 yr.

As you know, when you retreat a lift or level by robbing it, the adjacent grooves in those areas accumulate gases. Under certain atmospheric conditions these gases work down into the live workings of the lower levels.

Most of the fans used in ventilating such workings are of the force type and, of course, work very advantageously and effectively. We had one borehole that drew off so much gas that the vent pipe, which was somewhat over 20 ft in height, was set afire by youngsters shooting lighted arrows into its discharge. It would throw a flame 20 or 30 ft into the air.

C. Smith—The gentleman mentioned that these borehole drilling methods were being used in the west; this work is being done in West Virginia. I believe from something that was said that he did not follow me. If I may have half a moment, I would like to demonstrate it.

If you have an open cavern and your borehole comes in like this (*illustrating drawing on blackboard*), that is infinity. It is not necessary to pump it out 400 ft. By using a material such as would be fast setting, and which expands on setting, or by using a plastic, all of which are being currently used, a wall is built up. That is done by building the material, and there is an exact setting time. It may be 155 sec; you know that at 154 sec it is a fluid and at 155 sec, given a certain temperature, that material will immediately settle out.

C. Enzian—Under water?

C. Smith—Yes. You pump that ahead of the cement and follow with your cement. Consequently the cement has a proper grouting mixture. Let us say it is a 16 lb slurry. In that case you build your material into a wall at a set point, your mud being out here with an angle repose like that (*indicating drawing on blackboard*). Cement a vertical wall and then, after you reach and permit this to set, you prove your job by using high pressures to which I have referred. You know that if 2000 lb pressure is used and no break-through occurs, it is a good job.

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* Hazleton Pa.

A. L. Barrett*—What is to prevent the flow of water moving out?

C. Smith—This is built up with a very heavy slurry of mud which acts as a slow-moving mass and looks like the lava that comes out of a volcano. That merely acts as a mass to get your initial set. It is simple and easy. The size of the cavity, the interstitial measure, does not matter a bit. In every case, measured amounts of material are being used. Any assumption may be made. It could be 100 ft of clear hole into which you would drill, that is, a cavern of 100 ft, and it is perfectly practical to seal it off. I would not hesitate at all to say I could go to Mammoth Cavern and get a complete seal around the borehole.

L. A. Shipman†—I am awfully glad my friend insisted on a little further explanation. People in the oil business know how to drill holes and know grouting and certain methods much better than we in the coal business.

I did not think he made clear the fact that he starts out with low pressure and builds up a quick setting compound and gets his setting time before he goes to his high pressure. I am glad he was stubborn enough to repeat and emphasize this.

F. Sturges—Suppose we wanted to put down one 36-in. shaft, 150 ft in depth, what would be the cost?

C. Smith—In that case, it would be about $300. The cost of the job is a uniform one: $225 plus traveling time for the equipment which is heavily truck-mounted, plus the cost of the cement. I have given a rough figure for the maximum amount of cement that would be put in or used—about 1000 sacks for 580 ft.

F. Sturges—You believe that the whole job could be done for around $1,000?

C. Smith—That would be a high price. You have a vertical borehole and the first thing you do is put mud on the outside. In that case the cheapest method would be to use stove-pipe casings to get a very inexpensive method. In that way you would use ten sacks of cement, plus a job that either Mr. Halliburton or our company would do for $300.

C. Smith—Mr. Sturges, would you tell us how those boreholes are to be equipped and supplied with escape equipment?

F. Sturges—With an exhaust pressure fan at the foot of the shaft, to force the air up.

C. Smith—I may have missed the point. Will there be cage equipment for escape purposes?

F. Sturges—There will be.

C. Smith—Some of the members of the committee have expressed concern over the fact that there may be too much enthusiasm over the characteristics of small holes. That is a point about which apparently very little data have been developed. I know of one operator who thinks he is going to bleed the gas out of his seam, but we do not know yet how well that is going to work.

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* Pittsburgh Coal Co., Library, Pa.

† Southern Coal and Coke Co., Knoxville, Tenn.

The Action of Certain Microorganisms in Acid Mine Drainage*

By M. E. Hinkle,† Member AIME and W. A. Koehler‡

(New York Meeting, February 1948)

Introduction

The oxidation of pyrites and marcasite in coal-mine strata to produce discolored acid mine drainage has long been explained by chemical reactions occurring in three stages: 1. The iron sulphide minerals are oxidized by air in the presence of water to produce a nearly colorless solution of ferrous (iron) sulphate and some sulphuric acid. 2. The ferrous sulphate and part of the sulphuric acid combine with additional oxygen to form ferric (iron) sulphate. 3. The ferric sulphate hydrolyzes to form a reddish-brown precipitate and additional sulphuric acid.

Nevertheless, chemical data alone are not sufficient to explain all observed facts. The first stage seems to proceed more slowly in the chemical laboratory than in nature. The second and third stages normally occur to a marked extent within a few days with raw mine water. However, the visible discoloration that characterizes these latter stages will not develop for months if the mine water is sterilized.

By means of bacteriological techniques, two organisms have been isolated from acid coal-mine drainage. These are believed to have a part in promoting the chemical reactions summarized above.

One organism is considered to be *Thiobacillus thiooxidans*, which is known to be able to convert elementary sulphur or sodium thiosulphate to sulphuric acid. Its relation to the formation of acid mine drainage has not yet been established. Apparently it does not promote the oxidation of ferrous iron compounds in mine drainage to the ferric state.

The second organism has not been identified by name but is described as "Gram negative, non-spore-forming rods approximately 0.4 micron wide by 0.8 to 1.0 micron long." A pure culture of this when added to sterile mine water will cause the latter to become amber and develop the typical rusty precipitate. However, it has not been found to convert sulphur compounds to sulphuric acid.

Further research is planned to determine whether or not these microorganisms play a significant part in the first stage of oxidation whereby pyrites and marcasite minerals go into solution. Some of the conditions found in abandoned sections of coal mines, such as a reduced percentage of oxygen and increased percentage of carbon dioxide in the air, as well as the existing temperature and acidity ranges, will be tolerated by and may even be favorable to these organisms.

Acid Coal Mine Drainage

Acid coal mine drainage has been the concern of coal mine operators, industrialists, sportsmen and public officials for many years. The amount of acid flowing into our streams in the coal-producing areas varies considerably from mine to mine and from

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‡ Department of Chemical Engineering and Acting Director, Engineering Experiment Station, West Virginia University.

region to region and it is difficult to estimate even approximately the total amount. Herndon and Hodge[3] have made a study of it and give some estimates on the amount of acid produced. By no means is all coal mine drainage acid in nature; two mines only a few miles apart may have acid and alkaline drainages respectively.

In those mines in which acid is formed, ferrous sulphate is formed as a co-product. The ferrous sulphate oxidizes to the ferric stage and then hydrolyzes to form the ferric hydroxide, causing the reddish and brownish discolorations seen in the streams fed by acid mine drainage.

A fundamental study of the causes of acid formation in some mine waters has been authorized at West Virginia University under the sponsorship of Bituminous Coal Research, Inc. This research agency is supported by nearly 300 coal companies and associations, representing about 200,000,000 tons of annual coal production, and it sponsors various research projects of importance to the coal producer, the consumer and the public.

MINE DRAINAGE WATERS

The drainage from many mines is definitely acid in nature although that from some mines is alkaline. A survey of the hydrogen ion concentration of nineteen mines in Ohio, Pennsylvania and West Virginia showed a range from an acid mine water in Pennsylvania with a pH of 2.3 to an alkaline mine water in West Virginia with a pH of 8.7. In determining pH values of mine drainage waters close checks were obtained with a colormetric method, Beckman pH meter and Macbeth pH meter.

FORMATION OF ACID

The sulphuric acid formed in mines has its origin in the various sulphur-bearing materials associated with the coal, the most important of which is iron sulphide, occurring as pyrites and marcasite. These compounds appear as finely disseminated crystals in the coal or as bands parallel to the bedding plane of the coal, as flat lenses and as concretions commonly known as "sulphur balls." Some sulphur may also be present as sulphates and as organic sulphur.

As long as these sulphur compounds remain with the coal in unmined areas no acid is formed, but after removal of the coal so that the pyrites and marcasite remaining behind are exposed to air, acid is formed fairly rapidly. This is particularly true in the abandoned and unventilated sections of the mines. The oxidation of the sulphides to form sulphates and acid has generally been considered one of simple atmospheric oxidation:

$$2FeS_2 + 7O_2 + 2H_2O \rightarrow 2FeSO_4 + 2H_2SO_4 \quad [1]$$

The oxidation of pyrites is a slow process, however; in fact, it is difficult to produce any appreciable oxidation in the laboratory. Carpenter and Herndon[1] call attention to the slow oxidation of pyrites in the laboratory and suggest the possibility of anaerobic bacterial action as being responsible for the rapid oxidation of pyrites in the mines.

Colmer and Hinkle[2] in our laboratories studied numerous acid mine drainage waters for bacterial activity and give a detailed report on four typical samples. One sample was taken from a flume outside the mine, which received drainage by means of a pump from an abandoned area several hundred feet below the surface. A second sample was taken inside the mine as it ran out of an abandoned area, along the main haulage way. A third sample came directly from an abandoned area and was collected as it issued from a bore hole in the coal. A fourth sample was taken from a strip mine as the water issued from under a spoil bank.

All the samples were inoculated into the

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[a] References are at the end of the paper.

synthetic medium* of Waksman and Starkey[9] used for the culture of *Thiobacillus thiooxidans.* Growth appeared in all the samples within 7 to 10 days. Subcultures were then made into fresh media and repeated transfers always yielded an actual culture. Material from the flasks produced minute straw-yellow to cream-colored colonies when streaked on the surface of the solid medium of Waksman.

Growths taken from the liquid or solid substrata were morphologically similar to the published description of *Thiobacillus thiooxidans* and physiologically the organisms fit the description of the known sulphur oxidizer. They can secure their carbon dioxide from the air, their nitrogen from ammonium sulphate and their energy from the oxidation of elemental sulphur or thiosulphate. The pH of the liquid medium after one or two weeks of growth in the laboratory varied from 1.5 to 0.9 showing that acid is formed during the oxidation.

An analysis of the air from two of the abandoned areas from which water samples were taken showed a CO concentration of zero, CH_4, 12.47 and 18 pct respectively; CO_2, 3.6 and 3.8 pct respectively, and O_2, 3.2 and 9.4 pct respectively. According to Vogler, LePage and Umbreit[8] it can be expected that *Thiobacillus thiooxidans* would not find some of these gaseous components prohibitive to its action; in fact, they found a tendency to increased rates of sulphur oxidation at lower oxygen concentrations. Vogler[7] points out further that CO_2 may act as an oxidizing agent for sulphur. Starkey[6] called attention to the fact that *Thiobacillus thiooxidans* is able to grow when the medium contains 0.5 molal concentration of sulphuric acid.

The temperature of acid mine waters issuing from abandoned areas has been found in our investigation to range from 5 to 20°C. Vogler, LePage and Umbreit[8] found that bacterial sulphur oxidation proceeded over a temperature range of 7 to 45°C.

These observations point to the conclusion that possibly *Thiobacillus thiooxidans* or an organism very similar to it is an important factor in the formation of sulphuric acid in some, if not all, abandoned mines. Further study aims to determine, among other things, why, if this bacterium is an important factor, acid is not formed as readily in actively worked mines as in abandoned mines. This bacterium, however, is not responsible for the oxidation of ferrous iron to the ferric state mentioned in the next section.

Incomplete studies indicate so far that these microorganisms are not present in alkaline mine drainage.

OXIDATION OF FERROUS SULPHATE

Much more noticeable but not nearly as important economically is the oxidation of the ferrous sulphate to ferric sulphate and the subsequent hydrolysis to ferric hydroxide. The ferrous sulphate is a co-product formed with the sulphuric acid during the oxidation of pyrites and marcasite, as shown in Eq 1. The ferrous sulphate oxidizes to the ferric state, for which the reaction is commonly represented as:

$$4FeSO_4 + O_2 + 2H_2SO_4 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O \quad [2]$$

This has also commonly been considered as a process involving simple atmospheric oxidation. The ferric sulphate under suitable conditions hydrolyzes to form ferric hydroxide and more sulphuric acid:

$$Fe_2(SO_4)_3 + 6H_2O \rightarrow 2Fe(OH)_3 + 3H_2SO_4 \quad [3]$$

The acid drainage as it issues from the mine is often crystal clear, sometimes slightly turbid, sometimes of a slight amber color. Samples of such waters when first collected upon issuance from the abandoned mine contained the iron predominately in the ferrous condition. Upon storage in a glass jug, the clear water became turbid in a few hours, a reddish-brown precipitate usually appeared in one day and the iron was converted generally from the ferrous to the ferric state in from one to two weeks. Likewise, as the mine drainage water proceeds down a stream and becomes more dilute and therefore less acidic, the hydrolysis represented by reaction [3] takes place rapidly producing the brown precipitates in the stream.

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* Consisting of small amounts of ammonium, magnesium and iron sulphates, calcium chloride, potassium acid phosphate and powdered sulphur in distilled water.

In preparing a synthetic mine water made from laboratory reagents to approximately the same composition as a natural mine water, it was found that this synthetic mine water remained clear after standing exposed to air for months; no brown precipitate was formed. This gave a clue to the theory that some type of microorganism might be a factor in the oxidation. A natural mine water was then taken and divided into two parts in separate bottles. One part was first boiled, the other was unheated. The unheated sample formed the customary brown precipitate in a few days, while the boiled sample remained clear for weeks. Since the boiling could readily change the environment in the water, it was decided to try several bactericides. Colmer and Hinkle[2] added mercuric chloride, phenol, and formaldehyde respectively to clear acid mine water in concentrations ranging from one part in a thousand to one part in a hundred billion. Amber color developed only in those samples containing the smallest concentration of the bactericide in the control tubes. A residual concentration of one to a million of mercuric chloride prevented the amber color development. Phenol and formaldehyde prevented the change at one to a thousand and one to ten thousand respectively.

As a further check on possible bacterial action and to eliminate possibilities of the bactericidal chemicals mentioned above of changing the environment in the water other than destroying bacteria, a sample of clear mine water was filtered through an "E.K." germicide pad of a Seitz filter. This filter removes bacteria and produces sterile water. On exposure to air, the mine water so filtered developed no amber color.

As a further test a synthetic mine water, one that will not oxidize on standing exposed to air for months, was inoculated with a few milliliters per liter of fresh clear acid mine drainage. Oxidation developed in a few days as in the case of a natural mine water. Likewise, a Seitz filtered natural mine water, which will not oxidize on standing, when subjected to a few milliliters of an untreated natural mine water, oxidized in the usual manner.

These tests show that a bacterium has been found in acid mine drainage which is involved in the oxidation of ferrous to ferric sulphate, and since the ferric sulphate hydrolyzes spontaneously to the ferric hydroxide in the streams carrying the mine drainage water, the bacterium is indirectly a factor in the formation of the brownish precipitates found in many streams. It is, however, apparently not involved in the original oxidation of pyritic material to acid and ferrous sulphate.

Atmospheric oxygen, however, seems to be an important factor in the oxidation. If a bottle is filled with freshly collected natural mine water and then stoppered, no oxidation takes place. If a partially filled bottle has the oxygen replaced with CO_2 and is then stoppered, no oxidation takes place; but if the stopper is later removed the amber-colored precipitates gradually begin to form. However, the previously mentioned tests indicate that simple atmospheric oxidation alone will not produce the usual reddish-brown precipitate found in acid mine drainage waters.

To determine the nature of the agent or agents responsible for the oxidation, Colmer and Hinkle[2] inoculated 1 ml of the raw mine water which had just turned amber, into 10 ml of the same water which had been Seitz filtered. This water became amber and was then used as the inoculum for a second tube of the sterile water. After such serial transfers had served to enrich the agent or agents present in the colored water, a portion of the liquid from the last amber tube was streaked on the surface of a poured plate composed of equal volumes of sterile mine water and 3 pct agar-agar. After 6 to 10 days of incubation at room temperature at 22 to 28°C, an amber color appeared in the agar and very small colonies were apparent in and around the colored areas. Pure cultures of the bacteria were secured by repeated replating upon the acid water agar. Microscopical examination of the organism in the colonies proved the presence of Gram negative, non-spore forming rods approximately 0.4 micron wide by 0.8 to 1.0 micron long. When a pure culture of this bacterium was inoculated into either sterile raw acid mine water or synthetic acid mine water, that water became amber and developed the typical rusty precipitate, whereas uninoculated sterile control tubes of the same waters remained clear. The identity of the bacterium has not as yet been established, but it appears not to be *Thiobacillus thiooxidans.*

Acid mine drainage and the accompanying rust forming compounds in the streams have received much attention.[3,4] Hoffert[5] gives a brief review of various methods that have been proposed to counteract the acid issuing into the streams. Of the various methods proposed and tried, the mine-sealing program has been most effective. This sealing does not appreciably reduce the acidity of the water issuing from the mines but does decrease the volume and therefore the total acid. Further study is directed at West Virginia University under a new grant by Bituminous Coal Research, Inc. to investigate the true role of microorganisms in the formation of the acid and the iron oxidation products. It cannot, at this stage of the program, be predicted whether or not new and more effective measures to control acidity can be expected.

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DISCUSSION

(*G. H. Cady and H. J. Rose presiding*)

G. H. CADY—I wish to thank Dr. Koehler for this very interesting paper, and I am sure that he raised a number of questions which will excite some discussion.

H. F. HEBLEY*—Probably many in the room are wondering just what is the value of the work that has been presented by Dr. Koehler. Some of you may or may not know that there is a very strong agitation for the elimination of stream pollution throughout the whole Ohio Valley Basin.

In fact, there is a treaty between the various states of which there are nine involved, namely, New York, Pennsylvania, West Virginia, Virginia, Kentucky, Ohio, Indiana, Illinois and Tennessee; and a concerted drive is being made on all industrial wastes to prevent them from being discharged to the rivers.

States like New York and Pennsylvania have for decades used the streams for carrying away the industrial wastes as an easy method of getting rid of them. Packing houses, slaughter houses, cream factories, butter factories, paper mills and, last but not least, the acid mine water drainage that has been draining out of the mines into the stream system for years produce such wastes. You can see the magnitude of the problem with which we are faced.

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* Pittsburgh Coal Company.

I know that Dr. Maize as the Secretary for Mines of the State of Pennsylvania has a great deal to say at the present time as to what we can do with mine acid water.

It is only by the action of the mine acid water that some of the domestic sewage is dumped into the stream drainage system without any treatment whatever. That is all right as far as we are concerned, but somebody 50 or 100 miles downstream inherits the problem.

As long as the acid mine water is present the action of the bacteria is held in abeyance. As soon as the streams are diluted and travel far enough to permit putrefaction there are great islands of material made light by the gasification caused by the action of the bacteria under the stream which lifts these islands to the surface, thus releasing their gases. Then they subside once more.

Anyone who has lived in Philadelphia and knows the condition of the Schuylkill and the Delaware rivers at times has a wholesome appreciation of the matter. In our efforts to solve this problem of acid mine water—and it is a problem of great magnitude—with thousands of gallons a minute being pumped out, we often are told to "flocculate it and separate it out."

If we flocculate everything we would still have the material to handle and the quantity runs into tons. There is no market for it. Once we thought of making paint out of it. Based on our studies we could make enough of that material for paint in about a week to last the paint industry for three years.

In our effort to eliminate the mine acid water we have carried on work at the Bureau of Mines, at the Mellon Institute, and also at West Virginia University.

It was at West Virginia University, through the work of Hinkle and Koehler, that we stumbled on this possibility of an actual microorganism that is at work creating the mine acid water problem.

It may seem a rather restricted work that has been carried on, but the potentialities of it are vast. The work done at the Mellon Institute and the Bureau of Mines through its association with Bituminous Coal Research, Inc., is moving ahead deliberately, and I may say that in the last month we have made arrangements for the Mellon Institute to take samples every week from two different mines in Pennsylvania, one in the Pittsburgh seam and the other in Upper Freeport.

That work is yielding some rather enlightening information which we never even considered. For instance, we have found out that as far as the acidity is concerned the pH value does not mean as much as we thought. We have to take that with a certain amount of reservation; and it may start subsequent work.

I have taken quite a bit of time to outline the magnitude of this problem. We do obtain a measure of control by sealing some of the mines, but a lot of the mines have been abandoned and the owners cannot be found. Those aspects are all being looked into by the Mellon Institute. Possibly Dr. Maize may have something to add to it.

R. MAIZE*—I am a member of the Sanitary Water Board of the State of Pennsylvania. The Sanitary Water Board is making studies of all stream pollution, including industrial waste and acid mine water.

We have also notified the coal operators to keep the silt from the washeries out of the streams.

We are presently concentrating our efforts on the Schuylkill River where it is estimated there are 55 million tons of silt to be removed in order to put the river in such condition that it will be suitable for domestic use and recreational purposes.

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* Pennsylvania Department of Mines.

The coal companies in the anthracite region are spending a great deal of money to desilt the water from their mines before it is discharged into the rivers.

We are carrying on a very extensive program of mine sealing under the direction of the Department of Mines.

Our program is to seal all the mines in Pennsylvania which were abandoned prior to the passage of the Act; all mines abandoned after the passage of the Act, which became effective in September of 1947, will be sealed by the operators. The sealing, however, is to be done under the supervision of the Department of Mines.

In that way we will reduce the acid mine water flowing from mines that are now active.

There are thousands of mines in Pennsylvania which have to be sealed, and we are sealing them by authority of Act No. 490, passed in 1947.

I am not in a position to discuss the technical aspects of this program and the paper that was just read, but I am very much interested in getting the publications from the University of West Virginia because they are a great help to me in my studies of this problem.

One of our problems in sealing abandoned mines is the fact that we will go into an area where there are a lot of abandoned mines, but some operating mine has cut in or is adjacent to one of the abandoned mines.

Our aim is to seal those abandoned mines in that area. Then, as the operating mines are abandoned they will be sealed, and when the coal is worked out of that section we will eventually have the entire area sealed.

We are considering experimenting with the sealing portions of some presently operating mines. The Pittsburgh Coal Co. said that they would be willing to experiment by working sections or a panel in the mine; after it has been worked out, they will seal that area in order to ascertain whether the acid content coming from an operating mine could be reduced.

Of course that is a difficult problem with our method of mining. I understand that in Illinois they work their mines on the panel method, and when a panel is worked out they seal it. With this method of mining a large percentage of the coal is lost.

We have a method of mining in which we extract our pillars and whole areas are as one abandoned connecting gob; in those cases it is impossible to do any sealing in the mines until after the mines have been abandoned.

This briefly outlines our plans and problem. We are getting some results even in the short time we have been working under the new law.

The problem is such a gigantic one that it will take us many years to seal up the abandoned mines in Pennsylvania.

R. D. HALL*—For many years, I have studied the effect of microorganisms on the acidification of mine water, not with test tubes, or any like means, but by reading about, and mulling over, the subject. I have urged that if mine water is to be kept free of acid, the level of the water be kept as uniform as possible. If the water is allowed to rise, it reaches and dissolves the iron sulphates which have been formed by oxidation of the pyrite in the coal, fire clay or shales either by bacterial or inorganic action.

No iron sulphate can be formed under the water, for not enough air is present to permit such action, but certain oxidizing bacteria change the iron sulphates to microscopic iron sulphides, and these sulphides fall to the bottom of the water and coat the mine floor. As soon as the water falls low enough to expose these fine crystals of sulphide, they are oxidized and become sulphates, and the water dissolving them becomes extremely acid.

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* Forest Hills, L. I., N. Y.

This is what happens when a deep sump or even a shallower swag is divested of its water. Much corrosion occurs in pumps whenever such water is removed, though prior to the lowering of the water level, the water may not be unduly acid. Rapid dewatering may partly avoid this effect, for the iron sulphide may not change to iron sulphate fast enough to dissolve in the receding waters; but that has to be demonstrated.

In mines that produce a high-sulphur coal and have stood for some time, acid accumulations can be noted on the tops of the gobs at the sides of the roads. These sulphates are ready to dissolve as soon as the water reaches them. The rapid lowering of water is not as harmful as the raising of the water level, because the iron sulphide thus exposed may still be a sulphide when the water recedes and as it forms a sulphate only later, it may not be dissolved.

Perhaps the reason why the water from some mines, as those near Freeport, Pa., is alkaline is because the water entering such mines is sufficiently alkaline to inhibit the growth, proliferation and even the life of the microorganisms that turn the pyrite into iron sulphide. Bacteria, as is well known, are inhibited by an alkaline milieu. It is a very difficult matter to alkalinize all the water that may perhaps eventually enter a mine, but not so difficult to alkalinize the milieu of the pyrite exposed in the interior surfaces of the mine. Limestone rock dusting already does that, at least in part. Perhaps the rock dusting of gobs, and even of already wet swags, with limestone dust might prevent the formation of sulphates and at the same time, immunize the mines from coal-dust explosions though dust thrown on wet places will not serve for the latter purpose. Perhaps without knowing it, mine managements are already in a large degree circumventing much acidification.

It may be that what we need is automatic pumping with pumps that start whenever the water rises and do not attempt to remove the last drop. Intermittent pumping may save labor, load factor and, with large pumps, pump capacity, but it makes for intermittent service, the flooding of places and the formation of acid water and consequent corrosion. However, automaticity itself is not easily solved, and a lessened use of low-rate night current also may result.

W. R. CHEDSEY*—As a method of attacking this general problem there is the possibility of stopping the infecting microorganism. What brings this to mind particularly is a sudden recollection of some 20 or 25 years ago when Homer Turner, who is now dead, was at Lehigh University. He startled many of us by claiming to have found living bacteria in the heart of new coal; his procedure was such that there seemed to be no possibility of infection in handling after breaking the coal in the mine. That is, as though those were bacteria of the coal age, but still active. I was unable to get here early enough to hear the first part of the paper. Have you any information as to where this microorganism originates?

W. A. KOEHLER (authors' reply)—We do not have any such information at the present time, but Mr. Hinkle had raised the question as to whether that might not be a possibile source of these bacteria.

Meantime I would like to say something good for acid mine drainage. It seems that nothing is all bad. Some of the cities along the Ohio River get their city water from the river, and their system of treatment is dependent upon the acidity of the water caused by the mine drainage.

Some 15 years ago when the government started the program of mine sealing, and the acidity of the Ohio River was reduced, these cities could no longer purify their water by their usual method; the result was that the cities had to obtain waste pickle liquor from the steel mills and dump it into the water.

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* University of Illinois.

A. C. Fieldner*—The authors of this paper are to be congratulated on having made a new contribution on the mechanism of acid formation in coal mine water.

Bituminous Coal Research, Inc. deserves much credit for sponsoring fundamental research on a broad front by several different agencies. Different points of view are thus brought to bear on the problem and original contributions are likely to result from the work. This paper is an original contribution to our knowledge of the mechanism of acid formation at the source in coal mines.

We need to know more of the mechanism of acid formation in mines and in the further chemical changes that take place after the water issues from the mines and flows into streams. Dr. Maize has called attention to the sealing off of abandoned portions of coal mines. This is very important from the practical point of view. The engineer needs the results of fundamental research as a foundation for developing practical measures for reducing acid effluent from coal mines. He needs more knowledge of how to deal with the acid mine water problem at its very source. Such additional information will add to the progress already made in keeping water out of the mine or, where possible, in keeping abandoned mines sealed and thus preventing oxidation of pyrite and reducing the amount of acid discharged in the effluent from the mine.

H. J. Rose—I would like to continue with Dr. Fieldner's theme. The objective of the coal industry and the governor's sanitary board is the same, namely, to find a practical method of controlling the mine water problem; and in order to control it and prevent the acid mine drainage from forming, it seemed necessary to find out how it is formed. For a long time certain chemical explanations have been assumed to apply and they undoubtedly do apply to some extent. However, as research workers began to collect information systematically and to check the theories against the facts, there were many things that seemed to be partly contradictory and which the ordinary assumption did not explain adequately. That is why Bituminous Coal Research, Inc. felt that it was necessary to sponsor research on the subject.

As has been mentioned, we are sponsoring research at two places. At the Pittsburgh Station of the U. S. Bureau of Mines they are working on the chemical engineering approach as to how the objectionable compounds get in mine water. At West Virginia University, predominant attention has been given to the biological factors, and this study has already added to our knowledge.

The two main problems that are being considered, aside from the control itself, are as follows: First, how do these objectionable mineral constituents get in the water? That is, by what mechanisms do the solid minerals in the coal and associated strata change from an insoluble condition to water-soluble forms? Both the U. S. Bureau of Mines and West Virginia University project are working on that.

The second question is, what is the mechanism by which the relatively colorless mine waters containing dissolved ferrous iron compounds change to the highly colored ferric iron compounds with the liberation of additional acid?

There is no question in my mind but that the authors of the paper have shown conclusively that microorganisms play a part in the change of the nearly colorless ferrous iron solution to the colored ferric iron precipitate.

In further work, the authors will attack the problem as to how the material gets into solution in the first place. They expect that the biological method of attack will throw light on this subject.

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* U. S. Bureau of Mines.

We have to find a practical method of preventing or limiting the production of the soluble compounds in the water in the first place. As Mr. Hebley said, every chemical engineer who has investigated the problem thoroughly has agreed that it is impractical to treat acid mine water chemically. Many have tried that and all agree that the cost is prohibitive.

The authors would appreciate any comments from chemists or mining men as to practical observations that have been made with regard to conditions in the mine which seem to favor the formation of acid mine water.

It has usually been assumed that the presence of a large amount of atmospheric air favors the original formation of acid mine water. Some observations that the authors have made indicate that perhaps the strongest formation of soluble salts is in areas where the oxygen is reduced, that is, in the worked out inactive areas where there is only a small percentage of oxygen. In places with the best ventilation and the most air there appears to be little or no acid formed. The paper mentions that poorly ventilated conditions may be favorable to sulphur bacteria.

One conclusion to be drawn from the work to date is that we need to have more facts before suggesting a practical solution to the acid-mine-drainage problem.

W. A. KOEHLER—This work, as we see it, has three stages: 1. What causes the formation of the acid? 2. What can we do to destroy or nullify the acid or to stop its formation? 3. If we can not do that, can we turn it to some good instead of letting it do harm? We are still just scratching the surface of the first part of this program.

Shuttle-car Haulage

By David R. Mitchell,* J. H. Kelley,† Richard L. Ash,† J. D. Morgan, Jr.†, W. H. McCracken,† and William Bellano,‡ Members AIME

(Philadelphia Meeting, October 1946)

Rubber-tired shuttle cars are finding increased use as gathering units in underground mines. As the name implies, they shuttle back and forth between the loading machine and the section transportation system. In many mines their use has increased the efficiency of the loading and gathering operations, resulting in increased output per man employed and decreased costs per ton. A car motivated by cable reel is shown in Fig 1, and a car with battery power in Fig 2.

A brief chronological account of the development of rubber-tired haulage equipment follows:

1936. J. H. Fletcher[1] installed his first unit for the Blue Bird Coal Co., Carrier Mills, Illinois. This unit consisted of a battery-powered industrial tractor and a trailer with drop-bottom doors. The coal was transported for short distances to a hopper for discharge into mine cars.

1938. A battery-powered self-contained shuttle car with rubber-tired wheels was built by the Joy Manufacturing Co. Two units were installed for the Katherine Coal Co., Lumberport, West Virginia. The shuttle car consisted of a materials-receiving body with a conveyor in the bottom to move the coal forward during loading and to unload the coal at the discharge station.

1938. A cable-powered shuttle car with four-wheel drive and four-wheel steer was developed by A. L. Lee. Special wheels mounted on king pins on the car permitted all four wheels to be steered. The cable reel was driven by a torque motor to give the cable proper tension during the payout and uniform tension when rewound on the reel.

1940. A Joy shuttle car equipped with trolley poles and a cable reel was developed to operate on a double trolley-wire system.[2] This unit was installed at a potash mine of the International Minerals and Chemicals Corporation at Carlsbad, New Mexico. It is believed to have been the first installation of its kind, and the first of shuttle cars for handling material other than coal.

1941. Truck haulage[3] was introduced in the Tri State District by the Eagle-Picher Mining and Smelting Company. Storage battery electric trucks were rebuilt to handle a bottom-dump trailer.

1941—Joy shuttle cars were installed in a mine of the United States Gypsum Company, Oakfield, New York.

1943—Joy shuttle cars were installed in several bauxite mines.[4,5]

1945—Joy shuttle cars installed in a limestone mine.

Because of the nature of bituminous coal beds trackless equipment has had its greatest application in bituminous coal mining. Usually a shuttle car is filled by a continuous type of loader. In Fig 3 is shown a continuous machine loading coal into a battery-powered shuttle car, 48-in. coal. However, intermittent types of load-

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[1] References are at the end of the paper.

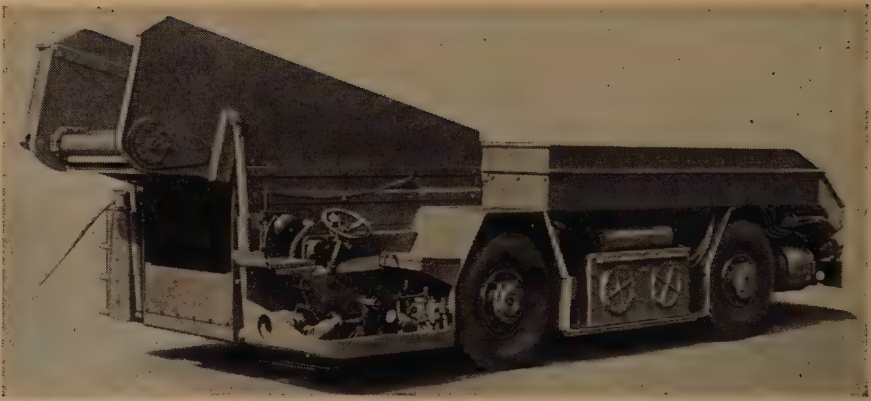


FIG 1—5SC CABLE-REEL SHUTTLE CAR, ADJUSTABLE ELEVATED DISCHARGE, HYDRAULIC STEERING. (*Courtesy Joy Manufacturing Co.*)

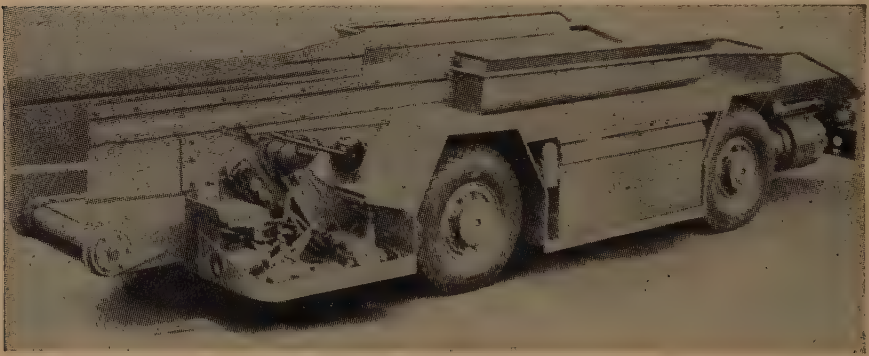


FIG 2—JOY 5SC BATTERY SHUTTLE CAR.

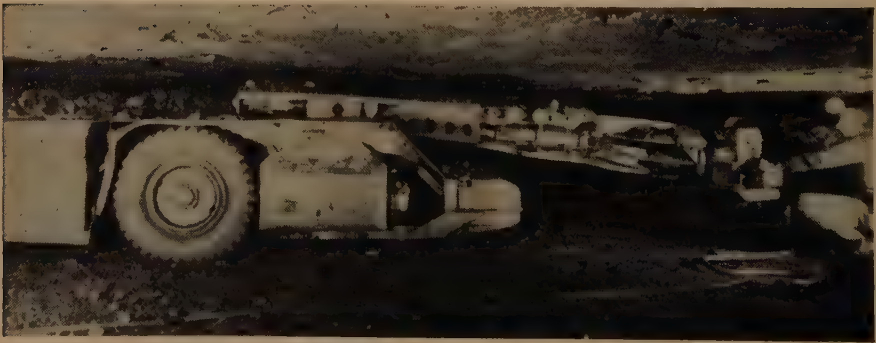


FIG 3—JOY LOADER AND SHUTTLE CAR, 48-INCH COAL.

ing machines can be used. No special equipment is required for shuttle cars to discharge onto conveyors. For discharging into mine cars an elevated discharge, an elevating conveyor (Fig 4), or a ramp is required. Cable-car installations require facilities for quickly repairing broken cables. Battery-powered cars require battery-changing and charging stations, as shown in Fig 5.

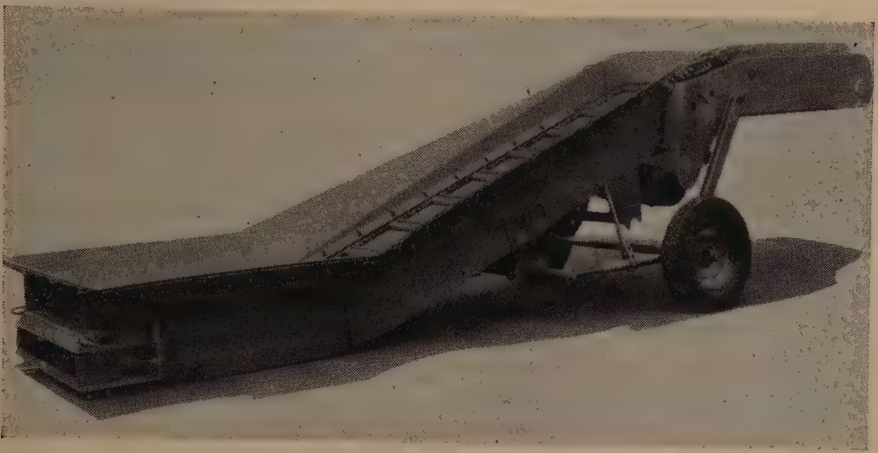


FIG 4—JOY PL11-7R ELEVATING CONVEYOR.

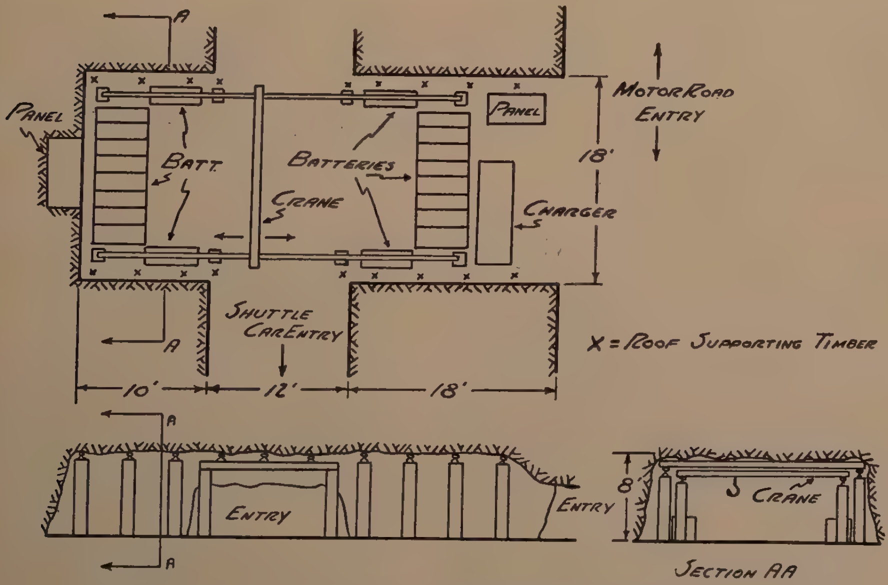


FIG 5—BATTERY-CHARGING ROOM SHOWING EQUIPMENT, ARRANGEMENT AND TIMBERING.

Although used primarily for gathering, the shuttle car is sometimes used as part of the haulage system. At International Minerals and Chemicals Corporation potash mine the combination trolley-pole, cable-reel cars travel distances as much as

2500 ft. The shuttle car is especially adapted to move material relatively short distances where the tonnage to be handled is not sufficient to warrant the installation of more permanent types of haulage or conveyor equipment. The shuttle car has been used or may be applied for: (1) gathering service in room-and-pillar and block systems of mining, (2) transporting material from stopes to transfer raises, (3) storage or waste underground, (4) filling of stopes and worked out areas, (5) tunnel driving, (6) handling supplies, (7) gathering in longwall mines.

Essentials for Shuttle-car Operation

The layout and operation of shuttle-car sections require a careful consideration of many factors. Practice has been developed by a method of trial and error on the part of most mine operators. A proper layout and plan of operation are necessary to ensure maximum efficiency and maximum production. The use of standard cost methods and engineering controls will aid in the increase of production and the reduction of operating expenses.

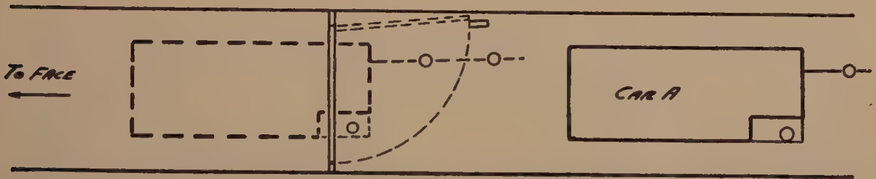
Mining Conditions

Up to the present time shuttle-car installations have been made in flat or slightly pitching mineral deposits. In all cases to date the room-and-pillar or block method, or some similar variation, has been used. The size and nature of shuttle cars limit their use to deposits or to mining systems where there is sufficient clearance to permit maneuvering the cars.

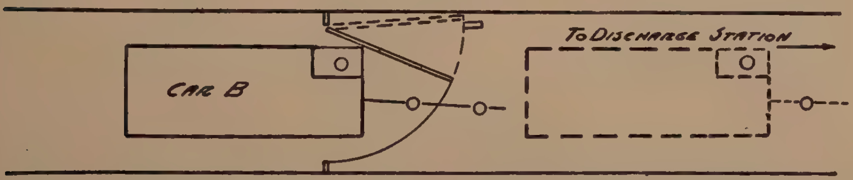
The physical factors to be considered, and comments pertaining thereto when considering installing shuttle cars, are:

1. Hardness and abrasiveness of the material to be mined.
2. Roof conditions:
    - *a.* Bad roof requires that provision be made to bypass uncontrollable areas, and that a rigid timbering plan be adopted and followed.
    - *b.* Dimensions of mine openings frequently are determined by roof conditions, affecting clearances and size of shuttle car that can be used.
3. Condition of the floor:
    - *a.* A soft, friable bottom is disadvantageous in shuttle-car operation, and time, labor, and materials used in maintaining a good roadway in these conditions may return dividends in the form of increased production.
    - *b.* A hard, dry bottom free of excessive grades is ideal.
    - *c.* Roadway construction needs much consideration by shuttle-car users. The introduction of a small electric-powered dozer may be the answer to quick drainage by ditching, and good, inexpensive roadway maintenance.
4. Pitch of the deposit:
    - *a.* Excessive grades are difficult to traverse; local pitching is even more difficult.
    - *b.* In pitching formations the shuttle-car cycle is slowed down and power costs go up. Wherever possible rooms and working places should be driven upgrade, so that travel of the loaded shuttle cars is downgrade.
5. Thickness of the deposit:
    - *a.* Thin seams permit little headroom and limit capacities of cars used; maneuverability of cars is important.
6. Presence of gas and dust:
    - *a.* In silica-bearing deposits, dust raised by shuttle cars is a physiological hazard. In coal mines there is an explosion hazard from gas and dust.
7. Removable impurities: In mines where relatively thick bands of bony, rock or other impurities can be stowed economically underground, the shuttle car can be used to move this material from the working faces to worked out areas.
8. Ventilation: Doors and other ventilation equipment interfere with shuttle-car operation. Cable cars particularly are not suited to travel through doors. Where two cars with complementary cable reels are used, cable interference occurs at the door and much time is lost by one of

the car drivers in opening and closing the door. The usual procedure for passing through a ventilation door where two cable cars are used is shown in Fig 6.



Car A, right-hand drive, proceeding to discharge station, pushes door open and permits it to close behind it. On return to face, car loses considerable time. Driver must leave seat to open door, then return to car and proceed nearly through opening, stop, and leave his seat a second time to release latch to close door.



Car B, left-hand drive, negotiates door with minimum of lost time. Driver need not leave seat to fasten or unfasten latch. Door remains open until car returns from discharge station.

Fig 6—Procedures of cable-reel shuttle cars for passing through ventilation door.

9. Other conditions: On many large-capacity (5 tons) cable-powered cars, too small a cable is being used. This leads to excessive cable failures and power consumption; for instance, the widely used No. 6 cable has a resistance of 0.4 ohm in 600 ft. At a normal load of 50 amp, a 20-volt drop occurs and 1000 watts of power are used in overcoming this resistance.

The new Federal Safety Code requiring three-conductor cable on rubber-tired equipment decreases the length of cable that can be placed on a reel and also increases the problem of cable repairs. About 100 ft less of three-conductor cable can be placed on the standard shuttle-car reel as compared with two-conductor cable. Cables may be flat (parallel) or concentric.

Except in the smaller shuttle cars, present manufacturers' design bases the capacity-limiting factors upon the size of tires and strength of the understructure. Thus, because the batteries weigh approximately ¾ ton more than the cable-reel equipment, the ratings of the large battery shuttle cars are about ¾ ton less than the corresponding cable-reel cars. Actual cubical contents may be 7 to 40 pct greater in the cable-reel car, allowing it to be overloaded. It has been found that frequently coal cars, when used to haul rock or coal containing a high percentage of rock, are carrying a load too large for the power of the motors, the size of the tires, and the size of the cable in cable-reel cars.

The cubical content of the two types of cars is the same for the small models, and it becomes physically impossible to load either type of car, designed for coal, above its rated capacity if it is carrying coal. If used for hauling rock cars can be overloaded.

To be successful, a battery-powdered shuttle-car operation requires a disciplined crew well trained in preventive mainte-

TABLE 1—*Summation of Conditions that Affect Selection of Shuttle-car Type*

Mining and Other Conditions	Favorable to Listed Shuttle Car Type	Remarks
Roof: bad	Battery	Freedom of travel—not restricted to one travel-way
Bottom:		
Wet and soft	Cable	If bad, not favorable to either type
Wet and hard	Either type	
Dry and soft	Either type	
Dry and hard	Battery	Advantage of speed
Heaving bottom	Battery	Freedom of travel
Inclination of seam; distance between synclines and pitch:		
Over 1000 ft:		
Over 8 pct.	Cable	
Under 8 pct.	Either type	
1000 to 500 ft:		
Over 6 pct.	Cable	
Under 6 pct.	Either type	
Less than 500 ft:		
Over 4 pct.	Cable	
Under 4 pct.	Either type	
Gas and dust:		
Gassy and dusty	Battery	Only type given a permissibility rating
Gassy and nondusty.	Battery	No performance advantage other than safety
Nongassy and dusty	Battery	
Nongassy and nondusty.	Either type	
Ventilation equipment:		
Doors	Battery	Cables interfere with doors and use of right-hand and left-hand drive incompatible with opening and closing
Brattice cloth	Battery	
Thickness of seam:		
High coal (over 6 ft)	Either type	
Medium (4 to 6 ft)	Either type	
Low to less than 4 ft.	Battery	Maneuverability of battery car important
Stowing or gobbing impurities underground.	Battery	No restriction on direction of travel
State or Federal laws and regulations:		
Permissibility	Battery	
Three-conductor cable.	Battery	Limits travel distance of cable car
Separate split of air required for changing and charging.	Cable	
Fireproof changing and charging stations	Cable	

nance and care of batteries. For cable-reel operations a crew trained to make quick cable repairs and possessing the ingenuity to plan adjustments of travel routes and passing points as adverse conditions are encountered is necessary.

Labor

The number of men in a shuttle-car section depends upon natural conditions, the mining method, and machines employed.

Some companies operate a production unit with as few men as possible and train them to do a number of different tasks. Men are seldom idle because of equipment breakdowns or other delays, since there is always work to be done, such as drilling holes, cutting, timbering, cleaning the face, cleaning the roadway, and at some operations gobbing. Examples are sec. 103 with 7½ men per shift and sec. 071 with 8 men per shift (see Table 4). Machine efficiency, based on the length of time machines are in operation, is usually low, and man efficiency is high. The men are all given a skilled classification and hence receive a high rate of pay.

Other companies operate shuttle-car production units with the idea of providing a man for each task. The aim is to keep the machines working a maximum number of hours per shift, obtaining a sufficiently high production to keep the tons per man-shift for the unit at a high level. Machine failures or other delays in the production cycle cause a number of men to be idle. Although mechanized breakdowns are costly in man-hours lost, ordinary tasks requiring little skill are done by labor at a lower wage scale than where a few highly paid specialists do all the work in the section. Up to 24 men are employed for one loading unit (see sec. 131 in Table 4).

In between these two extremes are found many arrangements. Cost per ton from the production unit is the deciding criterion. Crew size and allocation of specialists that work well at one mine are not necessarily applicable to similar mines.

The assignment of a mechanic to a mechanical mining section has met with varying success. The section mechanic is also expected to lubricate and inspect all equipment, aid in changing batteries and make on-the-spot repairs.

Many mines now use a central pool of maintenance men from which a man can be called to repair a disabled piece of equipment. These men work in the central repair shop when there is no breakdown in the section. The time lost while awaiting the arrival of a mechanic is a disadvantage of this system. Standby equipment is provided in some mines. This procedure lessens the immediate need of a mechanic in case of breakdowns.

Mechanical mining equipment, particularly shuttle cars, requires close supervision and control. Changing conditions require immediate decisions by a well trained foreman and crew. At many mines where shuttle cars are used, not enough attention is given to proper training of drivers. The breakdown of equipment or an unexpected natural condition can readily stop production, and unless the foreman has presence of mind man-hours will be lost. A foreman may be assigned two sections that are close together. In general, more than two sections cannot be supervised efficiently, and where natural conditions are bad, one section is sufficient.

SHUTTLE-CAR DESIGN

Shuttle-car dimensions range somewhat as follows: length, 19 to 24 ft; width, 7 to 7½ ft; wheel base, 7½ to 9 ft; height, 30 to 61 in.; capacity, 100 to 350 cu ft, level; turning radius, 22 ft.

Shuttle cars of 1946 model have centralized lubrication, four-wheel drive, and four-wheel steer with two-wheel steer optional.

Cars are built with a flat bottom, with a fixed elevated discharge, or with an adjustable elevated discharge.

Battery Cars

Battery-powered cars are equipped with two 48-cell lead-acid or nickel-iron storage batteries, one on each side of the car. These are 96-volt and range from 300 to 400 amp-hr capacity. Batteries last from ⅓ to 1½ shifts, depending on their condition, and care expended in maintenance. Battery abuses, such as wasting power, improper charging, and overdischarging, are common, and seriously affect battery life and performance.

Discharging batteries below the recommended minimum of 1.130 sp gr tends to build up positive and negative coatings on both plates of the cells. These coatings destroy the use of the portion of the plate they cover, and since it is of a permanent nature, the capacity of the battery is lessened considerably over a period of continued abuse.

The most recent innovation to the battery-car design is the series-parallel control, which eliminates the use of power-wasting resistances and permits the motors to develop greater torque when in series. Battery cars as now designed have manual steering.

The principal advantages of the battery shuttle car are its flexibility, greater speed, maneuverability, and permissibility. On account of its flexibility it is particularly adapted to handle waste and supplies; to bypass bad areas without serious loss of time; permit arranging standby positions close to the working face, thereby decreasing change-car time; to pass the other car and other equipment without cable entanglements; and to be operated without fear of cable blowouts. Its disadvantages are: charging stations are required, man-hours used for battery changing, and battery dead weight.

Cable-reel Cars

Cable-reel cars generally operate on 250-volt dc current supplied through No. 6 or No. 4, two-conductor or three-conductor cable, connected to a 250,000 CM feeder or 4/0 trolley wire. Cars can be equipped to operate on 500-volt current.

Cable-reel cars are equipped with hydraulic steering devices.

Recently a universal cable feeding device on the cable reel has improved the operation of this part of the machine. It permits the cable to trail in either direction, so that cable anchor points need not be placed near the discharge station.

Cable-reel car advantages are: less auxiliary equipment, no batteries to maintain, greater capacity of cars over 35 in. high, and a minimum of dead weight. Disadvantages are: decreased flexibility, man-hours and production lost by cable failures, and fire and ignition hazard from cable blowouts.

Operating Failures and Needed Improvements

Mechanical failures range from speed reducers to controller contactors. Manufacturers and mine personnel are constantly improving the design. Some of the most pressing needs for improved design are:

1. Larger size tires: (*a*) for better buoyancy in mud and water, (*b*) for better absorption of shock from rough roadways and quick stops.

2. The development of a device that will cut off the battery from the circuit, or display or sound some visual or audible signal when the batteries become discharged to a certain level. This would assist in enforcement of preventive maintenance of batteries.

3. Improvement of shuttle-car operator's comfort by better design of seats and addition of safety devices.

4. Redesign of cable, cable reel, cable-splicing equipment, and cable anchorages. Sluggish reels permit loose cable to accumulate on the floor, aggravating the whipping action when the car passes the anchor point and causing cable failures and pulled-out anchors. Frequently the cable pulls apart at former splices when splice clamps are used on flat cable. Most splicing is hasty and elementary. The splicer usually cuts a few of the copper strands while he is cutting off insulation, and the resulting splice becomes a point for overheating, sparking, and blowouts. It is likely that in the future three-conductor cable will be required in underground coal mines. In view of this, it is believed that the center wire could be strongly reenforced to give added strength and not greatly increase the bulk of the cable. Cable anchorages could be equipped with a spring attachment to absorb shocks.

Much improvement can be made on the feed of the cable reel, so that it will allow the cable to trail in three directions without friction—forward, rearward on one side of the reel, and rearward on the opposite side of the reel. Lack of this feature shortens the life of the cable, owing to frequent switching and the cables being run over by the shuttle car. Added to this feature could be a cable cleaner at the feed end to prevent dirt and mud from getting into the reel. This is particularly important where the bottom is soft and wet.

TABLE 2—*Cable Failures*

Month	Total Shuttle-car Shifts	Number of Cable Splices	Time Loss, Min
Nov. 1945	71	54	1,350
Dec. 1945	80	22	550
Jan. 1946	104	83	2,075
Feb. 1946	92	93	2,325

Average time loss per splice, 25 minutes.

Cable failures frequently are a cause of serious production loss. The record of failures and time loss at one mine operating a single cable car behind the loading machine in a production unit is given in Table 2. This is not average, neither is it uncommon. At another mine few cable failures occurred when one car was used but when two cars were used failures increased to an average of one every three shifts. Average time lost per splice in this mine was 28 min.

The significance of the cable blowout is sometimes misinterpreted in terms of time lost only, but this time cannot be compared with battery change time because like a roof fall, it is unpredictable. When a cable blowout occurs, the shuttle car is usually in a position where its disability disrupts the other shuttle car and the loading operation ceases. Often the cable breaks circuit without a blowout or other visual sign on the length of cable, and time is lost in finding the cable failure. Battery failures usually can be predicted and battery changes can be arranged so that little or no time is lost in breaking the production cycle. In some mines where batteries are changed during the shift, a shuttle car will change its batteries while the loading machine changes working place, and any gap will be assimilated by the other shuttle car. Several mines use the beginning or end of the shift for battery change.

The greatest hindrance to the use of shuttle cars is the inability to operate them in the soft and muddy bottom found in many mines. The design or development of some easily laid, economical roadway material might make it possible to use shuttle cars where the bottom is bad. Suggested materials that could be tried are: (1) asphalt matting, (2) pierced steel planking, (3) corduroy, and (4) timber-planked roadways. The last three have been used in some mines. Pierced steel plank like that used by the Army Air Forces proved unsuccessful at one mine.

Control of Shuttle-car Production Units

The mere installation of shuttle cars and their accessories does not in itself assure high productivity. Engineering planning and control are necessary to keep machines at a high level of performance. Deficiencies are best determined by time and method studies.

Time and Method Studies

Time studies are used to determine weaknesses in the production cycle. Method studies as applied in mining are used to determine the best procedure for routing and working machines to achieve maximum production. Judicious use of these two engineering tools provides factual data for improving operations.

Simultaneously with time studies, method studies should be made. These should include section layouts showing travel routes, passing and standby points, grades, distances, equipment locations, timbering, location of battery-changing and charging stations, and location of the main transportation system.

Other relevant information necessary for a complete analysis of the conditions and interpretation of results is a tabulation of kind and size of machines, personnel, and natural conditions

Table 3—*Legend of Time-study Symbols*

Symbol	Meaning
TE	Traveling empty
LZ̄	Loading time
TL	Traveling loaded
D	Discharging
WO	Waiting for other car(s)
WQ	Waiting for other equipment
HS	Handling supplies
DL	Delays while loading
DM	Delays due to mechanical reasons
DN	Delays due to natural conditions
DS	Delays at discharge station
DP	Delays from no preparation
DO	Delays due to all other reasons
PC	Delays from loader place change

Strategic placement of time-study men is necessary. Where there is sufficient height, one man can be assigned to a machine and follow it. Low headroom or long distances require the placement of a man at the face, one at the discharge station, and possibly one at such points of interference as passing stations. Usually two men suffice. The man at the face times specific operations of the loading machine and shuttle cars to which each time element applies. The man at the discharge station takes times applicable only to the shuttle cars. Where more than one time-study man is used, watches must be carefully synchronized. Time studies should be made during several shifts to avoid making erroneous conclusions resulting from abnormal conditions likely to occur on any one shift.

The time consumed by each machine for each separate maneuver and operation is determined. These times are then grouped into related items on a place or shift basis. For example, shuttle-car total trip time is composed of time attributed to productive effort, traveling empty and loaded, plus loading and discharging. All other times can be grouped into similar

TABLE 4—*Comparison of*

1	Section number						322	071
2	Deposit type	Potash[6]	Bauxite[4]	Bauxite[5]	Gypsum[7]	Salt[8]	Limestone	Coal
3	Natural conditions	Good	Good	Bad	Good	Good	Good	Med
4	Seam height	72 in.	84 in.	6 to 30 ft.	14 ft.	12 ft.	14 ft.	100 in.
5	Type of work	R & P	R & P	R & P	R & P	Room	Room	Entry
Equipment:								
6	Type shuttle car	42D9	42D11P	42E11P	42D9	42D9	60E3	Cable
7	Type loading machine	11 Bu	11 Bush	11 Bush	11 Bu	11 Bu	11 Bu	8 Bu
8	Type discharge station	El. con.	Belt	Belt	Raise	El. con.	Ramp	Ramp
9	Car-spotting method	Loco	Belt	Belt		Hoist	Loco	Hoist
10	Rated capacity, S C., ton.	8	5	5	10	8	12	7
11	Rated capacity, L. M., tons per min	4	4	4	5	4	6	1½
12	Number shuttle cars	2	2	2	1	2	2	1
13	Number loading machines	1	1	1	1	1	1	1
14	Number cutting machines	2				1		1
Labor:								
15	Loading and transportation	4	4	6	2	4	6	2
16	Cutting, drilling, and blasting	9		3	3	5	18	4
17	Supervision and maintenance	2		3		1	1	2
18	Miscellaneous	4	1	3	2	2		
19	Total labor	19	5	15	7	12	25	8
Performance:								
20	Maximum haul, ft	300	800	800	1000	750	500	470
21	Minimum haul, ft	50	100	70	500	100	100	100
22	Avg. tons per shift	600	200	150	350	500	640	140
23	Mine-car capacity, tons						8	3.3
24	Avg. tons per man-shift	31.5	40.0	10.0	50.0	41.7	25.6	17.5

1	Section number	243	421	232	234	233	111	260
2	Deposit type	Coal	Coal	Coal	Coal	Coal	Coal	Coal
3	Natural conditions	Bad	Good	Bad	Bad	Bad	Med	Med
4	Seam height	68 in.	66 in.	64 in.	64 in.	64 in.	64 in.	50 in.
5	Type of work	Entry	Entry	R & P	R & P	R & P	R & P	Room
Equipment:								
6	Type shuttle car	32D	5SC7XE 5SC7E	6SE	1-35D 1-6SCE	42D	2SC2D	42D6
7	Type loading machine	14 BU	11 BU	14 BU	14 BU	14 BU	8 BU	14 BU
8	Type discharge station	El. con.	El. con.	El. con.	El. con.	El. con.	Ramp	El. con.
9	Car-spotting method	Hoist	Loco	Loco	Hoist	Hoist	Loco	Hoist
10	Rated capacity, S. C., ton.	3	5	3½	3½	5	3½	6
11	Rated capacity, L. M., tons per min	5	4	5	5	5	1½	5
12	Number shuttle cars	3	3	2	2	2	2	2
13	Number loading machines	1	1	1	1	1	1	1
14	Number cutting machines	1	1	1	1	1	1	1
Labor:								
15	Loading and transportation	5	7	5	4	4	4	5
16	Cutting, drilling, and blasting	4	5	4	4	4	4	5½
17	Supervision and maintenance	2	6	2⅓	3½	3⅓	5	5
18	Miscellaneous	1						
19	Total labor	12	18	11⅓	11½	11⅓	13	15½
Performance:								
20	Maximum haul, ft	850	850	380	475	475	900	500
21	Minimum haul, ft	70	150	100	100	75	100	90
22	Avg. tons per shift	79	385	125	175	150	230	400
23	Mine-car capacity, tons	8.75	2.75	2.5	2.5	2.5	4.2	4.0
24	Avg. tons per man-shift	6.6	21.4	11.0	15.2	13.2	17.7	25.8

categories; such as delays due to natural conditions, delays due to failure of mechanical equipment, and delays occurring at the discharge station (Table 3)

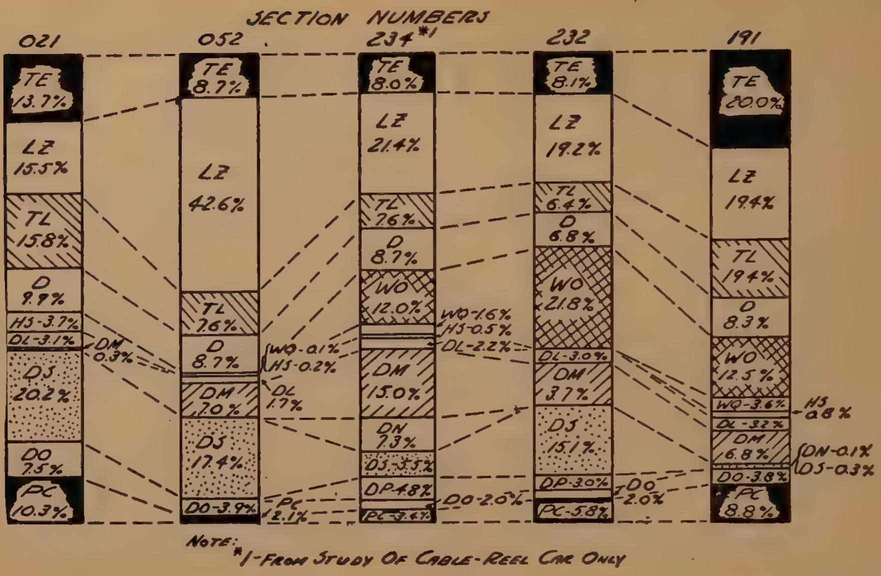
Case Studies

Studies are given of five production units utilizing: (1) single-battery car, (2) a single cable-reel car, (3) a combination of one battery car and one cable-reel car, (4) two cable-reel cars, and (5) two battery cars. In Fig 7 a representative time study is given of each one of these production units. Details as to equipment, labor and performance in these sections are given in Table 4. The legend of symbols used on

Shuttle-car Sections

072	131	101	102	103	092	091	241	242
Coal	Coal	Coal	Coal	Coal	Coal	Coal	Coal	Coal
Med	Med	Bad	Bad	Bad	Med	Med	Med	Good
100 in.	96 in.	72 in.	72 in.	72 in.	72 in.	72 in.	68 in.	68 in.
Pillar	Pillar	Room	Room	Entry	Entry	Room	Room	Room
42E	5SC-2PD	42D6	42D5	42E5	42D9	42E	32D	32D9P 32DP
8 Bu	11 Bu	7 Bu	7 Bu	7 Bu	14 Bu	14 Bu	11 Bu	14 Bu
Ramp	El. con.	El. con.	El. con.	El. con.	Belt	Belt	Ramp	Ramp
Loco	Hoist	Hoist	Hoist	Hoist	Belt	Belt	Hoist	Hoist
6	5	6	5	5	4½	6	3	3
1½	4	2	2	2	5	5	4	5
1	2	2	2	1	2	2	5	4
1	1	1	1	1	1	1	1	1
1	1	1	1	1	1	1	1	1
3	5	5	5	3	4	4	6	4
2½	5	4	4	4	4	4	4	4
1½	10	3	3	½	4½	4½	2	1
	4	1	1				2⅓	1
7	24	13	13	7½	12½	12½	14⅓	10
360	750	520	425	400	1100	400	930	900
100	450	150	100	75	100	100	100	100
165	450	360	325	160	275	325	230	155
3.3	7.4	6	6	6			8.75	8.75
23.6	18.8	27.7	25.0	21.3	22.0	26.0	16.1	15.5

051	052	191	193	192	021	761	153	152
Coal	Coal	Coal	Coal	Coal	Coal	Coal	Coal	Coal
Med	Med	Good	Good	Good	Bad	Good	Good	Good
44 in.	44 in.	43 in.	43 in.	43 in.	42 in.	42 in.	39 in.	39 in.
Room	Entry	Entry	Pillar	Room	Room	Entry	Entry	Entry & R & P
6SC-3XE	6SC-3XE	35D-9P	35D-9P	35D-9P	32D	32D-9P	Cable	Cable
14 Bu	14 Bu	14 Bu	14 Bu	14 Bu	14 Bu	14 Bu	14 Bu	14 Bu
Ramp	Ramp	Belt	Ramp	El. con.	El. con.	El. con.	Direct	Belt
Belt	Belt	Belt	Hoist	Hoist	Hoist	Hoist	Hoist	
3½	3½	3	3	3	3	3	3	3
5	5	5	5	5	5	5	5	5
1	1	2	2	4	1	2	1	2
1	1	1	1	2	1	1	1	1
1	1	1	1	2	1	1	1	1
2	2	4	4	9	4	5	2	4
4	4	4	3	6	4	4	4	4
1⅔	⅔	1	½	2½	3	2	2	2
½	1			1	1			1
8⅙	7⅔	9	7½	18½	12	11	8	11
280	350	460	800	870	600		350	350
120	140	100	100	100	100		60	60
165	130	150	100	500	110	175	120	220
3.1	3.1		3.2	3.2	2.0	2.0	3.0	3.0
20.2	16.9	16.7	13.3	27.0	9.2	15.9	15.0	20.0



a

	Section Numbers 021	052	234*1	232	191
Total Minutes In Shift	450	470	440	420	426
Type Of Work	Room	Entry	Rep	Rep	Entry
Mine Car Loads	55	44	63	51	59
Shuttle Car Loads	36	36	41 {22*2 19	32 {19*2 13	72 {39*2 33
Number And Type Shuttle Car	1-Batt.	1-C.R.	1-Batt 1-C.R.	2-C.R.	2 Batt.
Number Of Places Loaded	6	9	5	4	10
Labor	12	7⅔	11½	10½	9
Total Tons Produced	110	136	158	128	189
Tons Per Man Shift	9.2	17.7	13.7	12.1	21.0
Ratio Of Tons Produced To Average Tons	1.00	1.05	0.90	1.02	1.39

Notes: *1 From Study Of Cable Reel Car Only
*2 Car Studied

b

FIG 7—COMPARISON OF TIME-STUDY RESULTS FOR ONE SHUTTLE CAR PER SECTION STUDIED.

section layouts and other sketches is shown in Fig 8.

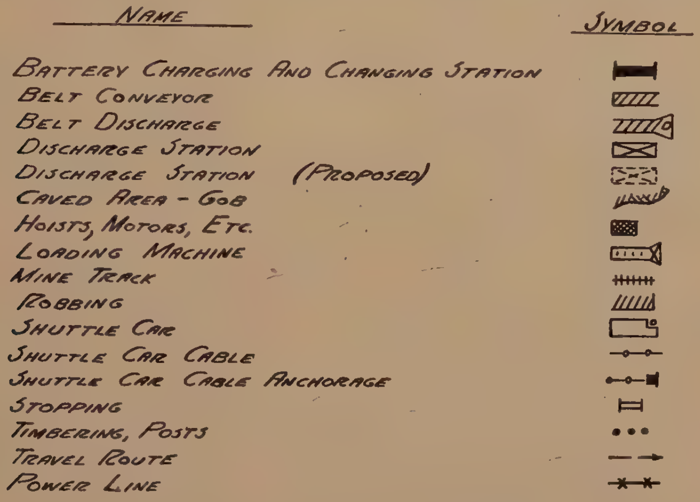


FIG 8—LEGEND OF SYMBOLS FOR SHUTTLE-CAR DIAGRAMS.

Section 021. *One Battery-type Shuttle Car per Loading Unit*—The section layout is given in Fig 9. Roof and floor conditions are extremely bad, necessitating erratic mining. The coal height is 42 in. The method of travel for the shuttle car to two different working faces is shown in Fig 10. Rooms and entries are driven 15 ft wide, which with timbering permits little clearance for the shuttle car. For the shift studied considerable time was lost through delays at the discharge station, primarily because of no mine cars. Of the cases given in Fig 7, this section was the only one requiring more time for traveling loaded than empty. This was due to poor travel ways. Despite excellent supervision, delays due to all other reasons (*DO*) are high, attributed to obsolescent auxiliary equipment and adverse natural conditions. Adverse travel-way conditions accounted for the large percentage of time lost during place changing (*PC*).

Section 052. *One Cable-reel Shuttle Car per Loading Unit.*—This operation is an example of entry driving in low coal (44 in.). A section layout is given in Fig 11. The discharge station placed alongside the conveyor obviously tended to cause the left entry to be favored, and necessitated an uneven advance. The change in location indicated in Fig 11 and shown in Fig 12 corrects this deficiency. Relocating the discharge station considerably improved the efficiency.

Poor face preparation caused an excessive amount of time to be used for loading. Travel times were low. The largest single delay occurred at the discharge station (*DS*), resulting from the lack of mine cars at the discharge of the belt conveyor. Delays because of equipment failure (*DM*) absorbed much time, the longest of which was a cable blowout requiring 27 min to splice. Other time losses were low.

Section 234. *Two Shuttle Cars per Loading Unit, One Cable-reel and One Battery Car.*—Only 90 pct of the average tonnage was produced during the shift studied. The coal height is 64 in. The section layout is given in Fig 13 and method of passing diagrams in Fig 14*a* and *b*. The time-study data apply to only the cable-reel shuttle car.

The largest single type of delay occurred because of mechanical failures (*DM*)

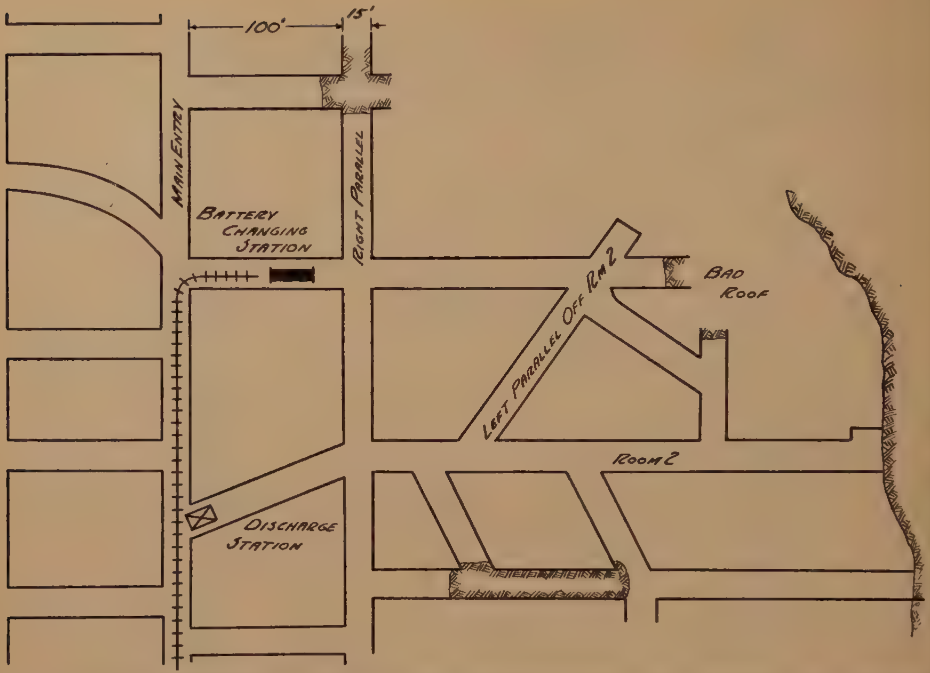


Fig 9—Room development, section layout, single-battery type shuttle car, section No. 021.

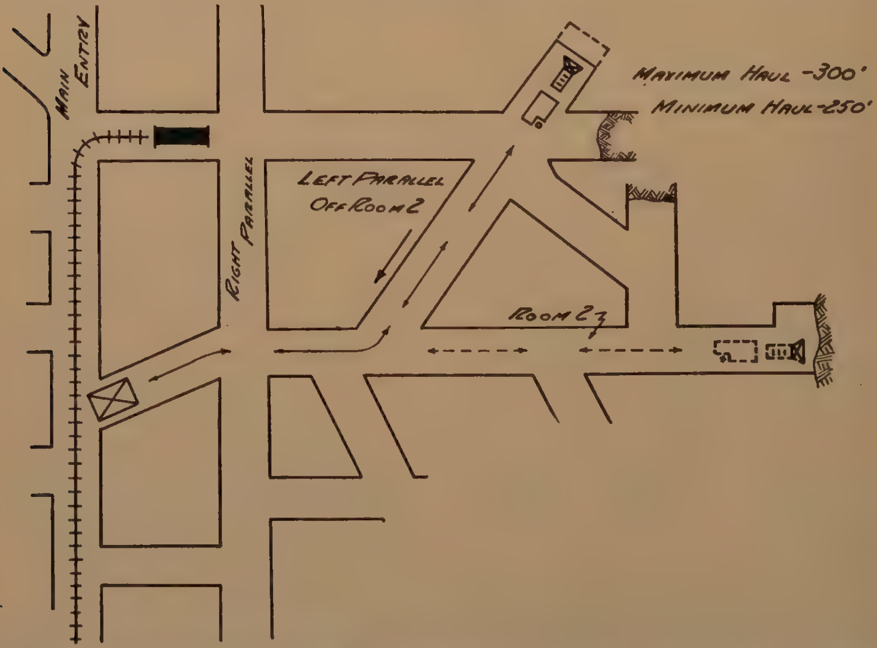


Fig 10—Room-development, method of travel, single-battery type shuttle car, section No. 021.

accounting for almost one hour of total shift time. This resulted from a combination of one cable blowout and a breakdown of the loading machine while changing to another place. The time lost because of natural conditions (*DN*) was caused by poor travel ways.

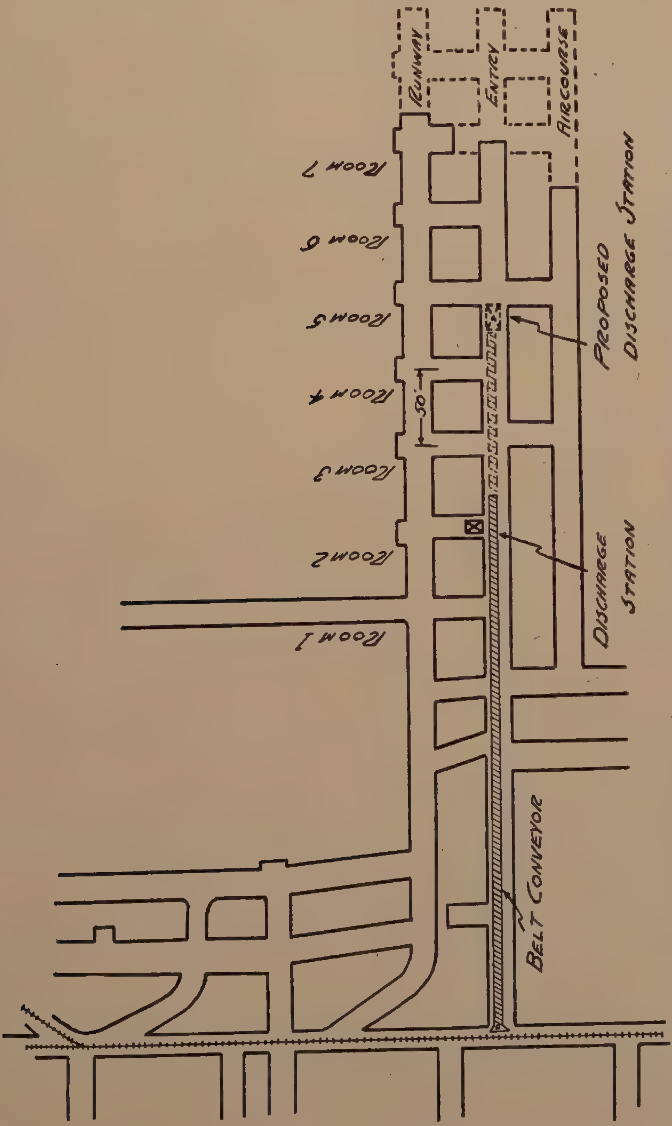


Fig 11—Entry driving, section layout, single cable-reel shuttle car, section 052.

The time loss resulting from waiting could be lowered considerably by using hangers when loading out entry pillars as shown by Fig 14*b*, and in Fig 14*a* when loading in rooms 27 and 28. Use of hangers would make possible the placement of standby positions nearer places of loading and prevent the battery-powered car from running over the cable of the cable-reel car. Moving the discharge station to the left main entry would reduce distances and eliminate considerable maneuvering.

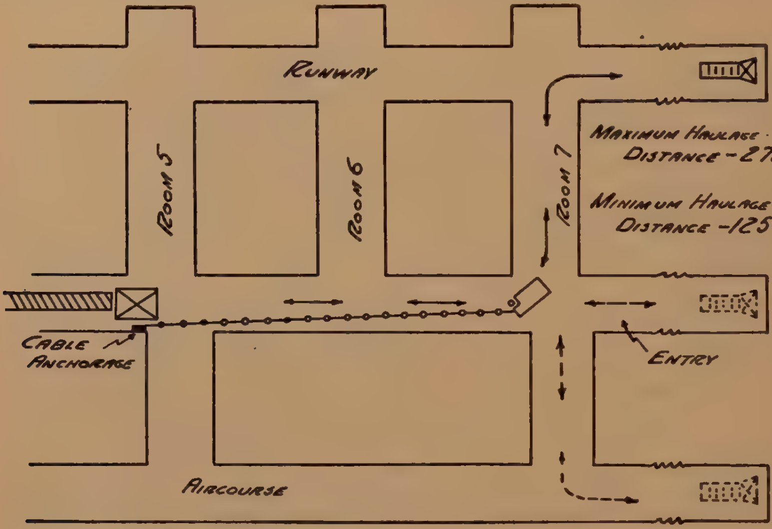

Runway
Room 5
Room 6
Room 7
Maximum Haulage Distance - 275'
Minimum Haulage Distance - 125'
Cable Anchorage
Entry
Aircourse

FIG 12—ENTRY DRIVING, SINGLE CABLE-REEL SHUTTLE CAR METHOD OF TRAVEL, SECTION NO. 052.

Section 232. Two Cable-reel Shuttle Cars per Loading Unit.—The section layout for this room-and-pillar operation is given in Fig 15. Two cable-reel cars are being used in 64-in. coal. One car has left-hand drive; the other has right-hand. Methods of passing are given in Fig 16.

The time study applies to the right-hand drive car. Production was normal. Considerable time was lost waiting for left-hand drive car to be loaded (*WO*). Waiting time could be reduced when loader is in room 35 by having right-hand car standby in a crosscut between rooms 34 and 35 and use room 34 for travel. Left-hand car would standby, outby crosscut in room 35 and use room 35 for travel. Delays occurring at discharge station (*DS*) were predominantly because of no empties and to some extent by car spotting. The large delay listed as due to mechanical equipment (*DM*) was caused by failure of a loading-machine cable.

Section 191. Two Battery-type Shuttle Cars per Loading Unit.—This is an example of entry driving in 43-in. coal. The section layout and methods of passing are given in Fig 17, 18*a* and 18*b*, respectively. The largest delays are waiting for the other car (*WO*), and delays due to mechanical reasons (*DM*). The latter occurred from battery changing. Place-change-time consumed more time than in any of the other cases studied. It is to be noted that placement of the discharge station at the head of the belt conveyor would decrease travel time to and from left parallel face. This would permit less frequent moving of the discharge station and should result in less time lost by waiting.

Bauxite and Potash Mines

Examples of section layouts using two shuttle cars per loading unit in bauxite deposits[4,5] are given in Fig 19 and Fig 20. Methods of passing at the Republic bauxite mine are shown in Fig 21*a* and 21*b*. The use of shuttle cars in a potash mine[6] is shown in Fig 22. Performance information for these sections appears in Table 4.

The system of cable hanging of the shuttle-car unit in the Republic bauxite mine is unique in that it represents use of

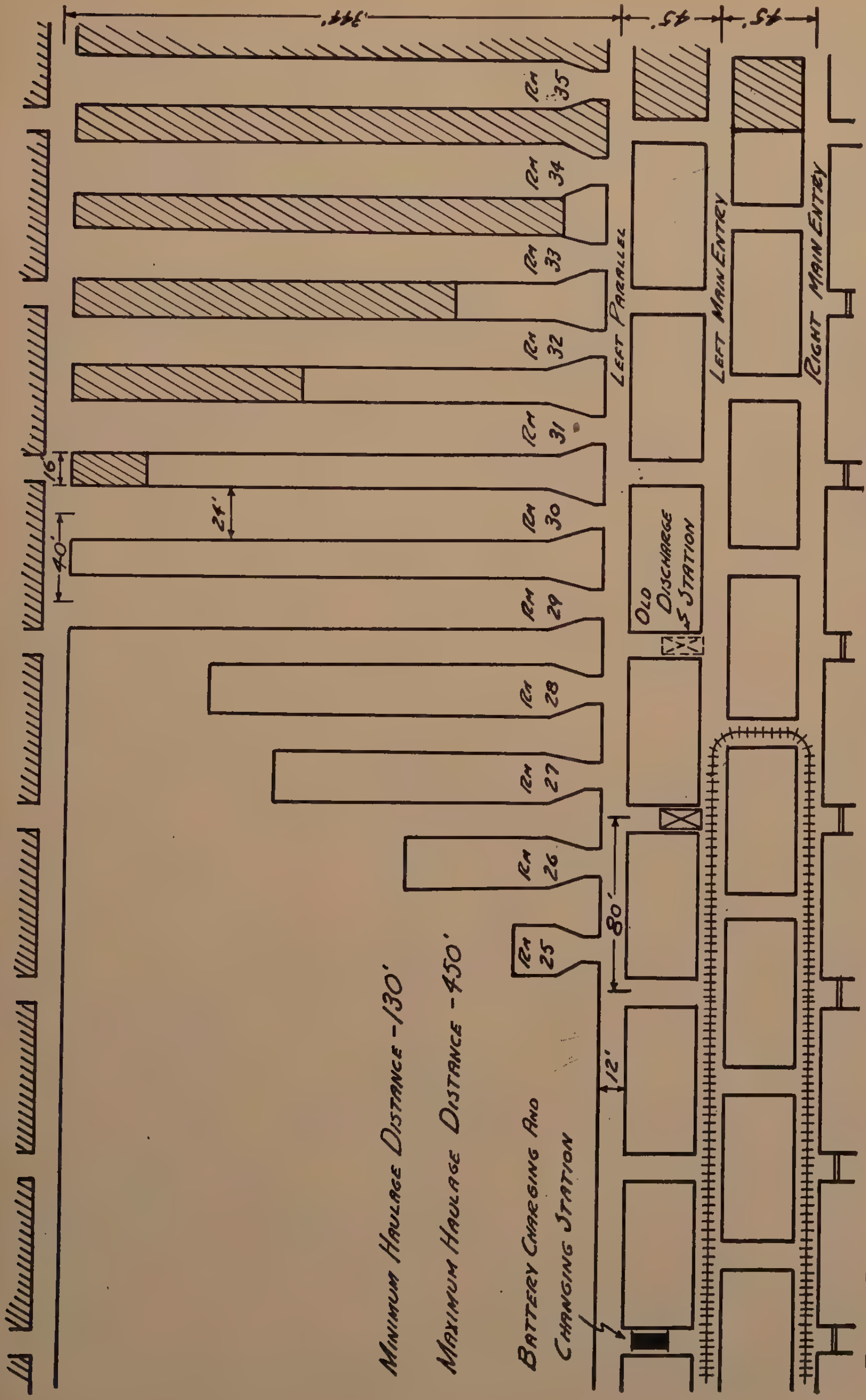
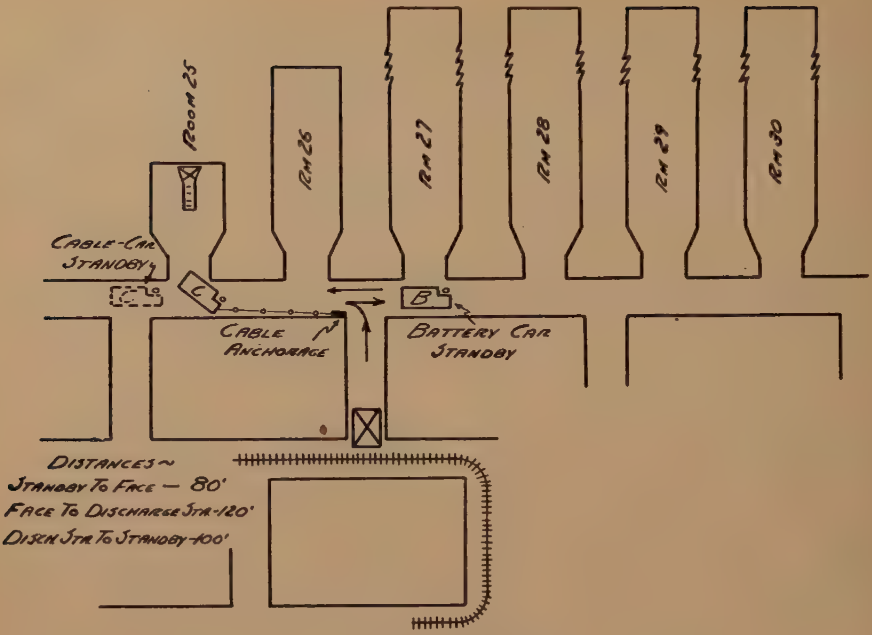
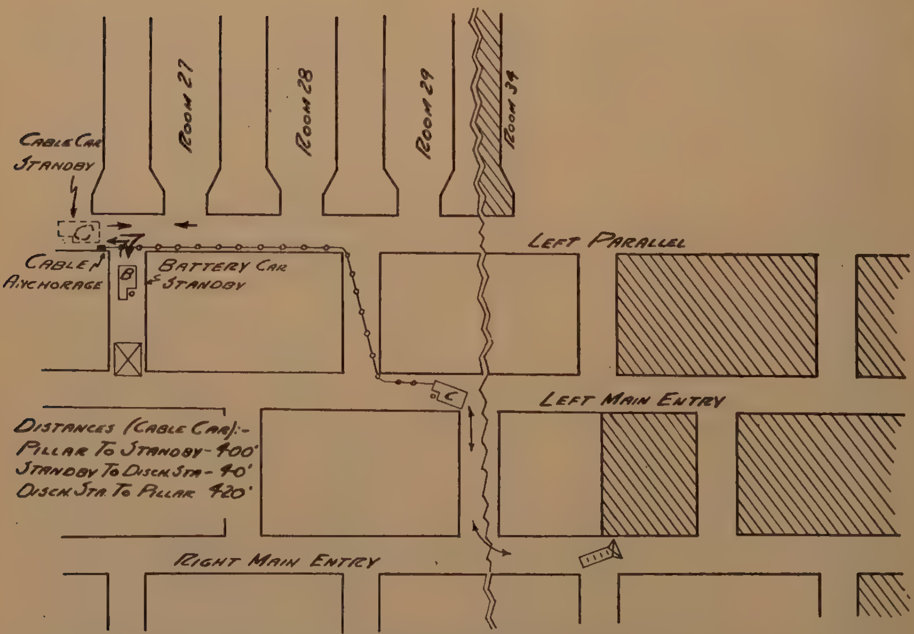


FIG 13—ROOM-AND-PILLAR WORK, SECTION LAYOUT, ONE CABLE-REEL SHUTTLE CAR AND ONE BATTERY-TYPE SHUTTLE CAR, SECTION NO. 234.



a. Room Development.



b. Pillar Recovery.

FIG 14—ROOM-AND-PILLAR WORK, METHOD OF PASSING ONE CABLE-REEL SHUTTLE CAR AND ONE BATTERY-TYPE SHUTTLE CAR, SECTION NO. 234.

cable hangers and spool guides on shuttle cars. The size of the drift gives small travel clearances, necessitating special arrangement for passing. One car must park in the passing station while the other car is traversing the drift.

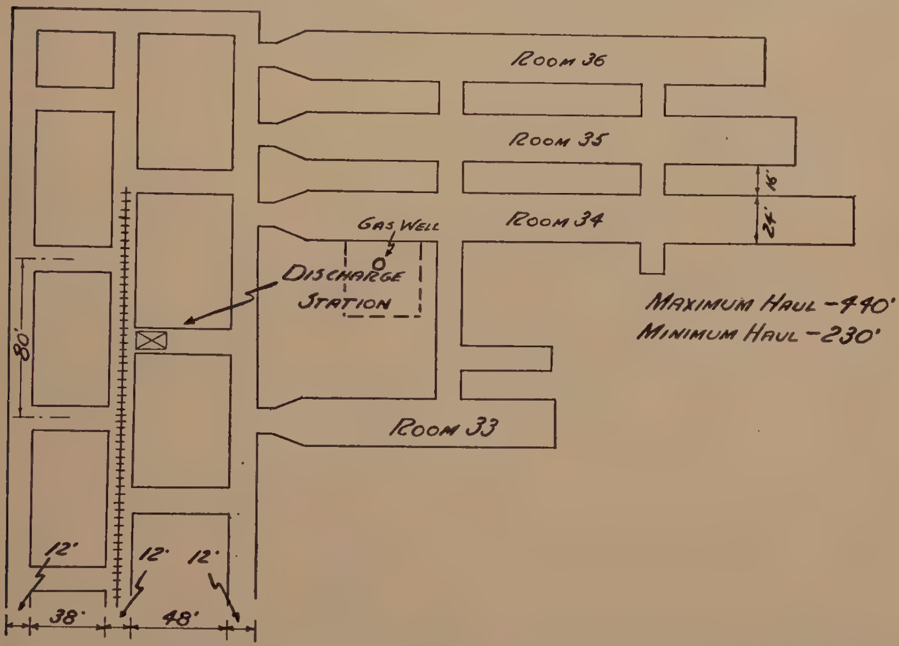


FIG 15—ROOM-AND-PILLAR WORK, SECTION LAYOUT, TWO CABLE-REEL SHUTTLE CARS, ONE LEFT-HAND DRIVE AND ONE RIGHT-HAND DRIVE, SECTION NO. 232.

PERFORMANCE

Production in tons per man-shift is the best means of indicating operating efficiency in underground mining. Many factors may contribute to low production per man-shift, such as improper selection and utilization of equipment, unskilled personnel, improperly designed mining systems, and adverse mining conditions. Method and layout studies are important. A number of layouts and accompanying performance records are given in the 1946 *Mechannual.*[9]

Production performances for 32 shuttle-car sections at 20 different mines are given in Table 4. Where the first two figures are the same in the section numbers, it means that all sections listed under this number are in the same mine. For instance, sections 071 and 072 are sections 1 and 2 in mine number 07. In section 111 a remote-controlled locomotive is used to spot mine cars. In section 153, bottom is taken in entries and shuttle cars dump direct into mine cars.

The type shuttle-car and loading machine are given by manufacturer's number. All of those listed are manufactured by the Joy Manufacturing Co., except for the units listed as "cable." Battery-powered cars are indicated by the presence of the letter *D* and cable-reel powered cars by the letter *E* in the number giving the type shuttle car.

All production figures, items 22 and 24, are averages of several weeks duration. Shift production often varies widely from the average; for example, at mine 10 during a month's operations of 23 shifts, the range in mine cars loaded per shift by sections follows: section 101, 35–57 mine cars, average 44.4; section 107, 37–59, average 48; and section 103, 10–32, average 23.6.

Non-coal mines are listed first. Up to 50 tons per man-shift is being obtained. The low figure of 10 tons per man-shift at one of the bauxite mines is due to a very bad bottom condition, requiring the laying of corduroy in the travel ways.

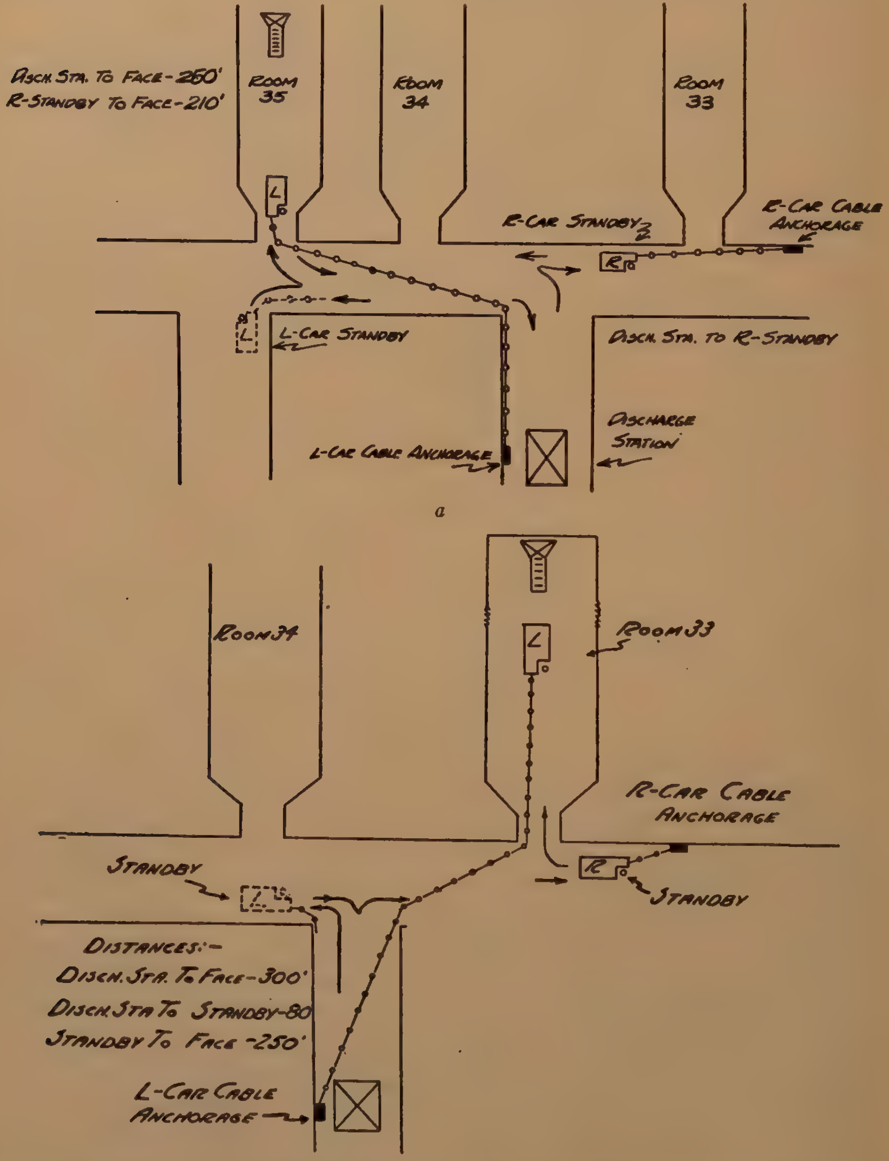


Fig 16—Room-and-pillar work, method of passing two cable-reel shuttle cars, section No. 232.

Coal-mine sections are given in order of seam height. Of the eight examples in high coal (72 in. and above) the best average production per man-shift occurred in room-development work where two battery-type shuttle cars were used per loading machine (sections 101, 102). This production is being achieved despite adverse mining conditions and the use of a small-capacity loading machine. However, three fourths of the sections produced more than 20 tons per man-shift, showing the suitability of shuttle cars in high coal for all types of work.

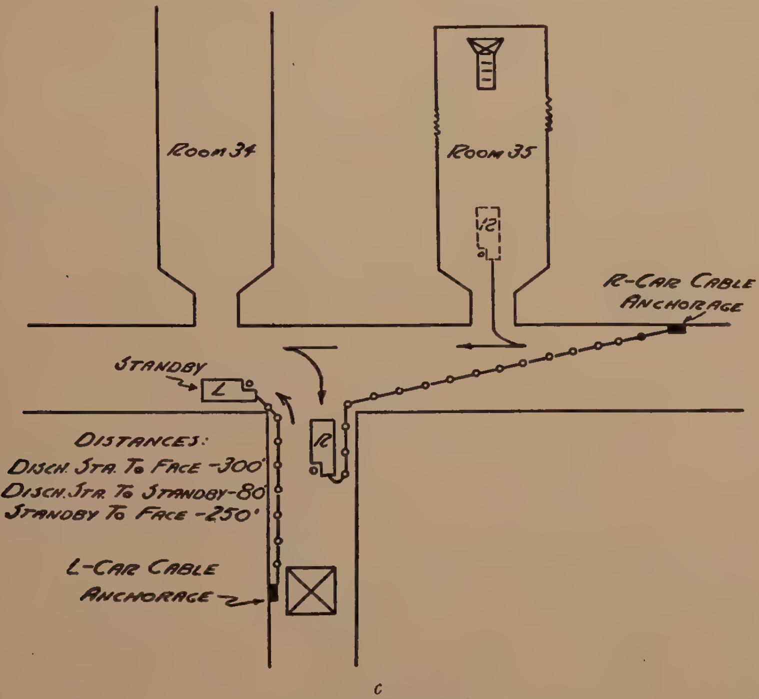


FIG 16 (CONTINUED).

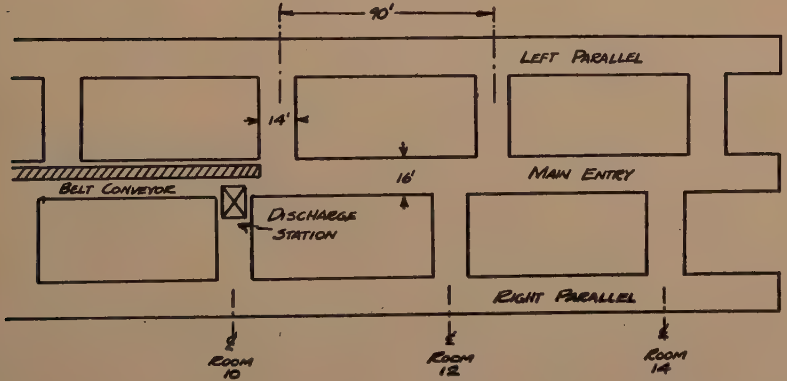


FIG 17—ENTRY DRIVING, SECTION LAYOUT, TWO BATTERY-TYPE SHUTTLE CARS, SECTION NO. 191

For medium height coal (71 to 48 in., inclusive), there are eight examples. Only two sections had an average production exceeding 20 tons per man-shift, one using three cable-reel shuttle cars per loading machine in entry driving (section 421) and the other using two battery-type shuttle cars per loading machine in room development (section 260). The latter example had the highest production in medium height coal with 25.8 tons per man-shift.

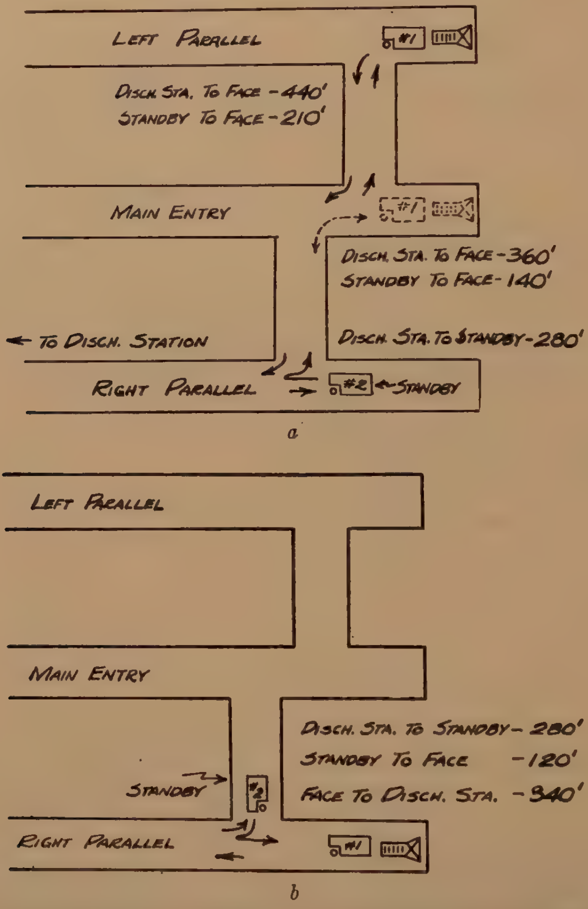


FIG 18—ENTRY DRIVING, METHOD OF TRAVEL, TWO BATTERY-TYPE SHUTTLE CARS, SECTION NO. 191.

Shuttle cars produced equally well in low coal (less than 48 in.) as in medium height coal. The highest tons per man-shift occurred in room development (section 192) where two units, each employing two battery-type shuttle cars per loading unit, discharged into mine cars at a common point. The units actually worked in separate sections with a maximum haul of 870 ft from the loading point to the discharge station. Of the nine examples given in Table 4 of shuttle-car sections in low coal, only one gave low production in room-development work (section 021), and that was on account of bad roof and soft bottom.

Considering a mine or a section of a mine as a whole, there are two methods of attack to get an overall high production per man-shift from all production units. One is to lay out the mine section so that the maximum haul is kept low and provide for each production unit about the same

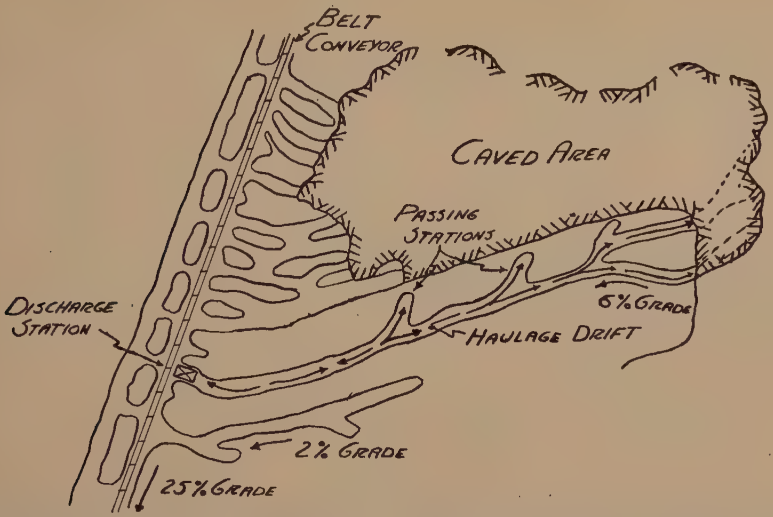


Fig 19—Shuttle-car layout for Republic bauxite mine, Republic Mining and Manufacturing Company, Bauxite, Arkansas.

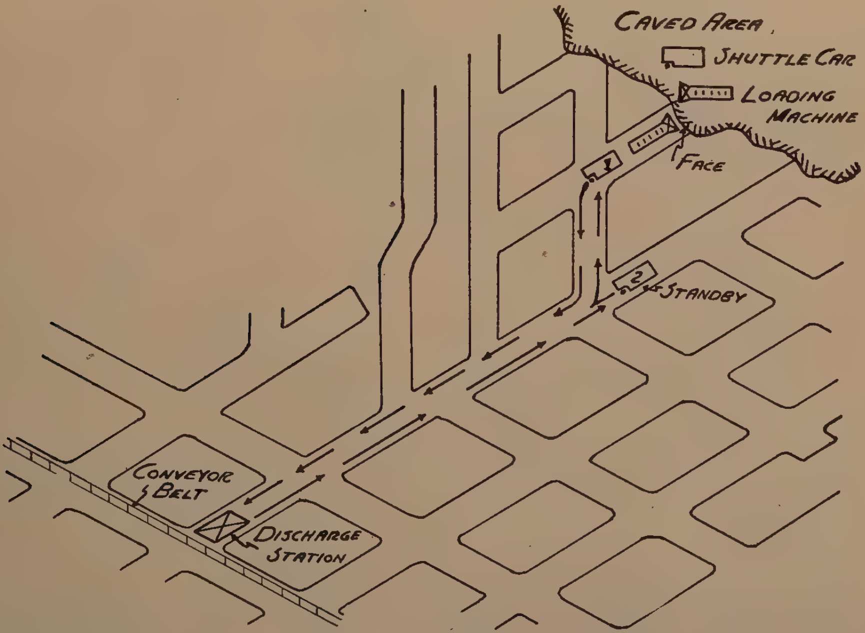


Fig 20—Shuttle-car operation in bauxite at Hurricane Creek mine of Reynolds Mining Corporation, Brooks, Arkansas.

duty whether on entry driving, room development, or pillar extraction. This procedure is frequently followed for cable-reel installations. Mines 05 and 15 are examples giving an average overall production per man-shift of 18.5 and 17.5 tons, respectively. The other method is to lay out the sections so that most of the coal is won and conditions made favorable for a high production from one phase of the operation, usually either room development or pillar recovery. Mine 19 in which battery cars are used is an example: 60 pct of the production comes from rooms, 20 pct from entry driving, and 20 pct from pillars, resulting in an overall average production of 23.9 tons per man-shift. This is an unusually high average production for low coal and particularly so since the maximum haul is 870 ft.

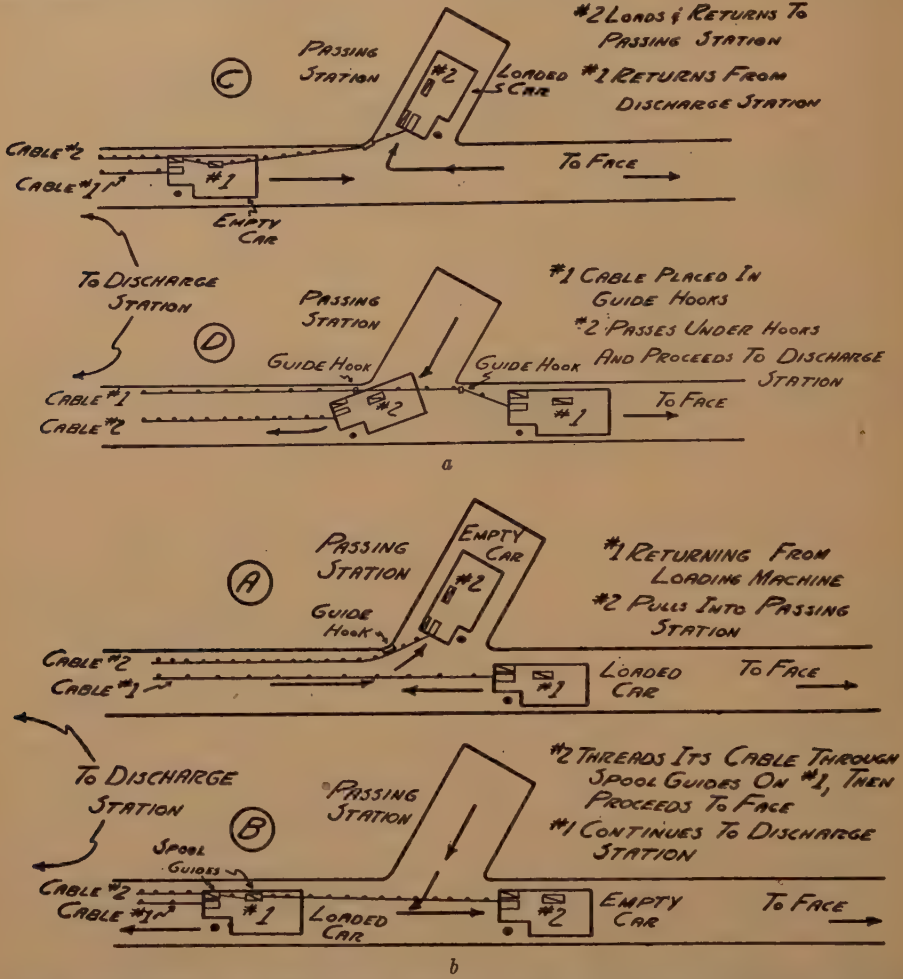

C
Passing Station
#2 Loads & Returns To Passing Station
Loaded Car
#1 Returns From Discharge Station
#2
#1
Cable #2
Cable #1
Empty Car
To Face
To Discharge Station
D
Passing Station
#1 Cable Placed In Guide Hooks
#2 Passes Under Hooks And Proceeds To Discharge Station
Guide Hook
Guide Hook
Cable #1
Cable #2
#2
#1
To Face
a
A
Passing Station
Empty Car
#1 Returning From Loading Machine
#2 Pulls Into Passing Station
Guide Hook
#2
Cable #2
Cable #1
#1
Loaded Car
To Face
To Discharge Station
B
Passing Station
#2 Threads Its Cable Through Spool Guides On #1, Then Proceeds To Face
#1 Continues To Discharge Station
Spool Guides
Cable #2
Cable #1
#1
Loaded Car
#2
Empty Car
To Face
b

Fig 21—Sequence of operations, showing method of passing without entangling cables for two cable-reel shuttle cars in drifts of Republic bauxite mine, Bauxite, Arkansas.

There are few instances where a direct comparison can be made between battery-powered shuttle cars and cable-reel cars in the same mine. At many mines where both types of cars are in use, the cable-reel car will be used in the sections that are favorable to its use and the battery car in sections favorable to its use.

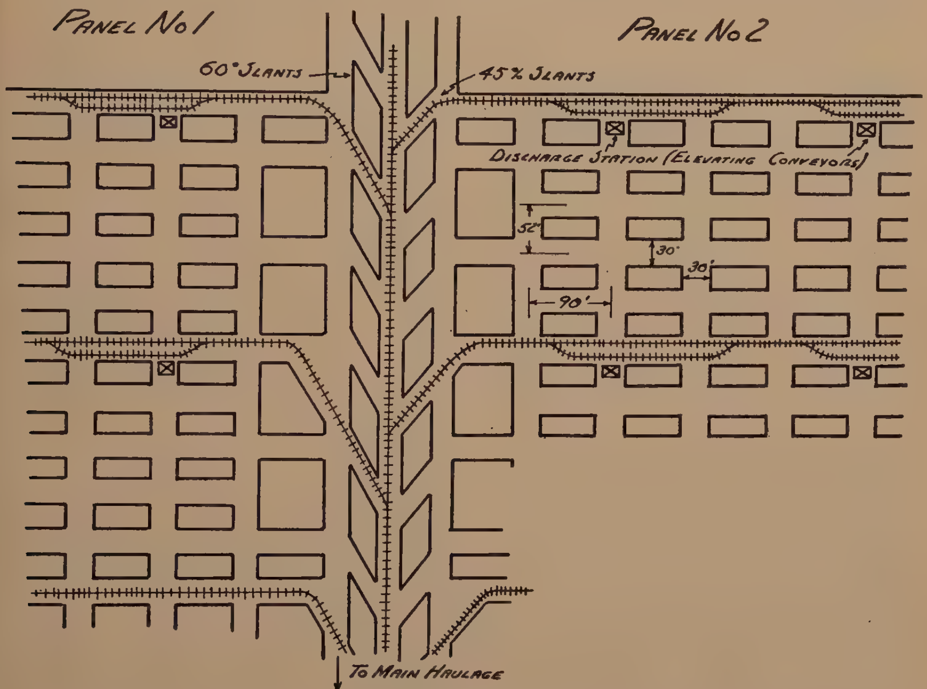


FIG 22—SHUTTLE-CAR LAYOUT FOR MINE OF POTASH COMPANY OF AMERICA, CARLSBAD, NEW MEXICO.

A direct comparison can be made at mine No. 23. For the conditions prevailing at this mine a combination of one battery car and one cable car (section 234) gave the highest and most consistent performance. Two battery cars (section 233) were next and two cable cars (section 232) lowest. These sections were all on room-and-pillar work with conditions about the same except that the all battery-car section and the combination battery-car, cable-reel car section had 100 ft greater maximum haul distance.

At mine 09 the cable-reel section in room work shows a higher average production than the battery section on entry driving. However, the battery section has a much greater haul distance than the cable-reel section. At mine 10 the opposite is true, with the battery sections in room work giving a higher average production per man-shift than cable cars in an entry section.

The number of shuttle cars used per loading machine varies widely. In low coal production, units employing one cable-reel car per loading machine are common. In many low-coal mines it has been found that when using two cable-reel cars production per man-shift dropped to about 70 pct of that of a single cable-reel production unit. At other mines in medium and high coal, production per man-shift has frequently been increased by using two and sometimes three cable-reel cars to a loading machine. Up to four and five battery cars have been used with one loading machine (see sections 241 and 242).

The number of shuttle cars to use per loading machine depends on the layout and natural conditions. Machine operating efficiencies based on percentage of total time a machine is doing productive work vary from 25 to 50 pct for loading machines and from 25 to 75 pct for shuttle cars. At mine 10, sections 102 and 103, the average operating efficiencies and other pertinent information are as shown in Table 5.

TABLE 5—*Pertinent Data*

Data	Rooms, Two Battery Cars	Entries, One Cable-reel Car
Shuttle-car efficiency, pct	56.4	65.5
Loading-machine efficiency, pct	46.5	37.3
Shuttle-car trips per place	10.5	4.4
Calculated shuttle-car capacity, tons	5.0	4.7
Shuttle-car speed, fpm	307	283
Loading time per shuttle-car trip, min	3.5	4.8
Discharge time per shuttle-car trip, min	0.65	0.85
Change-car time per shuttle-car trip, min	3.2	5.2
Load-out time per place, min	63.2	40.6
Average tons per man-shift	25.0	21.3

At mine 05 machine efficiencies varied as follows:

	Room Development, Pct	Pillar Recovery, Pct	Entry Driving, Pct
Shuttle-car efficiency	62–65	44–74	69 74
Loading-machine efficiency	33–43	25–45	37–44

COST

Cost per ton of coal produced is the deciding factor in selecting mining machinery provided the equipment under consideration meets required safety standards. Generally speaking, where natural conditions are favorable, shuttle-car production units frequently show a cost advantage over other methods of mining and gathering.

Of the various items entering into the total cost in coal mining, cost for labor is by far the most important item. Cost for labor ranges from 55 to 65 pct of the total cost in most of the mechanized underground bituminous coal mines in the United States. Cost of supplies is next, averaging about 20 pct of the total cost.

Since labor is the largest single item in the cost of producing a ton of bituminous coal at underground mines, increasing the productivity of labor provides the best opportunity for lowering total costs. An increase in the tons produced per man-shift usually indicates a corresponding reduction in the total cost of mining a ton of coal.

TABLE 6—*Assumed Conditions, Hypothetical Mine*

Location	Pennsylvania
Height of coal, ft	6
Roof condition	Fair
Bottom condition	Dry
Dust	Very little
Gas	No
Impurities separately handled	None (cleaning plant on surface)
Inclination of seam, pct	0–2
Number of shifts	2 per day; 50 per month
Length of working shift, min	480
Length of shift, portal-to-portal, min	540
Mine capacity, tons in two shifts	2500
Number of sections	5
Average section production, tons per shift	250
Cost of power, total mine; per ton	$0.10
Power used in section, kw-hr per shift	300
Number of men in section	13
Average distances, ft:	
Substation to battery station (central)	500
Substation to discharge station	1,500
Battery station to discharge station	1,000
Power distribution:	
Positive	1500 ft of 4/0 trolley wire parallel with 1500 ft of a 250,000 cm feeder
Negative	1500 ft of 60-lb rail single bonded

There is much discussion about the comparative costs of production units using the two types of shuttle cars. For comparative purposes a hypothetical mine is taken where either type of shuttle car can be used and where there is no marked advantage for either type. The assumed conditions are given in Table 6.

Estimates of capital and cost of labor are on the basis of prices and rates for labor prevailing during the fall of 1946. Maintenance and replacement costs are the averages of 25 mines studied. Power cost for the mine is assumed to be $0.10 per ton of coal produced.

Capital and Maintenance Costs

For simplicity, costs pertaining to the battery-charging station and for cable replacement are listed as part of the first cost of the equipment instead of replacement cost. This is done for convenience of comparing the cost of the items characteristic of a cable-reel shuttle-car installation with that of a battery-powered installation over a period of five years. It is considered that a cable-reel car will use one 600-ft length of cable at $0.66 per ft each 5 months of operation and the two cars will use 25 lengths in the 5-year period. It is likewise considered that postwar batteries will last 5 years and that the 3,500,000 sq ft to be mined will need the relocation of the central battery station three times in the 5-year period. It is also assumed that the charging station will serve 5 sections. The required equipment is given in Table 7.

TABLE 7—*Equipment Required*

Cable-reel Section	Battery Section
1 loading machine, caterpillar mounted, 5 tons per min rated capacity	1 loading machine, caterpillar mounted, 5 tons per min rated capacity
2 shuttle cars, cable-reel type, four-wheel drive and steer	2 shuttle cars, battery type, four-wheel drive and steer
1 cutting machine, universal, caterpillar mounted	1 cutting machine, universal, caterpillar mounted
1 elevating conveyor	1 elevating conveyor
1 mine-car hoist	1 mine-car hoist
1 electric drill, auger type, post mounted	1 electric drill, auger type, post mounted
1 air compressor	4 sets storage batteries, 350 amp-hr capacity
15,000 ft electric cable, No. 4, three-conductor	⅕ cost of moves of changing and charging stations
	⅕ battery-charging equipment including air compressor
Cost of equipment......... $69,750	Cost of equipment........ $66,650
Life of equipment 5 years	Life of equipment 5 years
Unit cost..$0.093 per ton	Unit cost. $0.089 per ton

Maintenance cost for the section over the 5-year life, including labor and supplies, is estimated at $142,500 for the battery section and $138,500 for the cable-reel section. Unit cost for maintenance is $0.190 per ton for the battery section and $0.185 per ton for the cable-reel section. The difference is largely the cost of one battery-station attendant per shift minus the cost of repairing one cable splice per two shuttle-car shifts.

It is assumed that each section will have the same average production, 250 tons per shift, considering the balancing effect of the greater speed and flexibility of the battery car on one hand and the greater capacity of the cable-reel car on the other.

Power

Ten cents per ton is taken as the power cost for the entire mine. Considering the mine equipped with cable-car sections, it is estimated that 60 pct of the power consumed in the mine is used to operate such nonfluctuating equipment as pumps, fans, and lighting, and 40 pct for equipment having a fluctuating demand. If it is assumed that the total number of kilowatt-hours used in one month's period is nonfluctuating, and that the demand for each 15-min period is constant for the month, the kilowatt billing demand factor is constant and the lowest possible. Calculating the cost for such an optimum rate of consumption, it is determined that although the nonfluctuating power is 60 pct of the total amount, it accounts for only 34.15 pct of the total cost of power for the mine.

The fluctuating power is estimated to be divided as follows: main haulage, 15 pct, and section production, 25 pct. Thus the main haulage accounts for 24.5 pct of the total cost of power and the section, 41.2 pct. Unit cost per ton for a cable-car section is

$$41.2 \text{ pct} \div 100 \times \$0.10 = \$0.0412 \text{ per ton.}$$

Total power consumed in a cable-reel section is calculated as follows:

Two shuttle cars = 51.5 kw-hr per shift
One 53-hp loading machine operating 125 min = 82.4 kw-hr per shift
One 60-hp cutting machine operating 200 min = 149.4 kw-hr per shift
Miscellaneous equipment and losses = 16.7 kw-hr per shift
Total, 300 kw-hr per shift.

The cost of power in the cable-reel section for one shift's operations is

$$\$0.0412 \times 250 = \$10.30/\text{shift}.$$

Then the unit cost per kilowatt-hour is

$$\$10.30 \div 300 \text{ kw-hr} = \$0.0343 \text{ per kw-hr}$$

The total power used in the mine is 300 kw-hr per section × 5 sections ÷ (25 pct × 100) = 6000 kw-hr per shift. Of this amount 60 pct, 3600 kw-hr, is nonfluctuating. The cost of this power is 34.15 pct of the total mine power cost, or

$$\$0.3415 \times \$125.00 = \$42.69 \text{ per shift}.$$

Then the average unit cost per kilowatt-hour of the nonfluctuating power is

$$\$42.69 \div 3600 = \$0.0118/\text{kw-hr}.$$

In Table 8 the difference in actual kilowatt-hours consumed by the two types of shuttle cars is calculated.

TABLE 8—*Estimated Power Consumed, Two Battery vs. Two Cable Cars*

Battery	Cable
4 traction motors, 7½ hp ea., operating 20 pct of shift	4 traction motors, 7½ hp ea., operating 20 pct of shift
2 conveyor motors, 7½ hp 2 ea., operating 14.5 pct of shift = 51.5 kw-hr.	2 conveyor motors, 7½ hp ea., operating 14.5 pct of shift = 51.5 kw-hr.
Battery charging and discharging losses, efficiency 65 pct = 27.7 kw-hr.	Controller resistances and cable-reel motor = 4.1 kw-hr.
Battery transportation, 2000 ft per shift at 300 ft per min., or 6.67 min. of shuttle-car travel = 3.8 kw-hr.	Line power losses and cable losses, 250 volts at section or 290 volts at substation = 8.9 kw-hr.
Total.... 83.0 kw-hr per shift	64.5 kw-hr per shift

The power cost for operating two cable cars per shift equals

$$64.5 \times \$0.0342 \div 250 = \$0.0088 \text{ per ton}.$$

For operating two battery cars, assuming all power used is nonfluctuating, the cost is

$$83 \times \$0.0118 \div 250 = \$0.0039 \text{ per ton}.$$

This is a difference of $0.0049. Section cost for a battery-car installation is thus estimated to be

$$\$0.041 - \$0.0049 = \$0.036 \text{ per ton}.$$

Labor

It is assumed that 13 men are used in each section as follows: loading, 2 men; cutting 2 men; shuttle-car operation, 2 men; drilling, 2 men; blasting, 1 man; discharge station, 1 man; timbering, 1 man; maintenance, 1 man; and supervision, 1 man. The total section-labor cost for one shift's operation, with two ½-hr man-trips per shift, is calculated to be $156.628 = $0.62651/ton.

Supplies directly consumed by the section are estimated to cost $0.20 per ton.

Items of labor and supplies do not contain the costs of labor and supplies included under maintenance.

TABLE 9—*Summarization of Section Costs, Battery vs. Cable-car Production Units*

Data	Cable Reel Cost		Battery Cost	
	Per Ton	Pct	Per Ton	Pct
First cost, equipment only, 5-year write-off period................	$0.080	6.69	$0.076	6.38
Cost of equipment characteristics to type (batteries, chargers, cable).	$0.013	1.09	$0.013	1.09
Maintenance on all section equipment.......	$0.185	15.47	$0.190	15.94
Power consumption.....	$0.041	3.43	$0.036	3.02
Labor (incl. section foreman).................	$0.627	52.42	$0.627	52.60
Supplies................	$0.200	16.72	$0.200	16.78
Amortization, and/or fees, royalties, and taxes.................	$0.050	4.18	$0.050	4.19
Totals.............	$1.196		$1.192	

A summarization of the calculated and estimated costs is given in Table 9.

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Modern Haulage to Meet Local Conditions

By G. S. Jenkins,* Member AIME

(Philadelphia Meeting, October 1946)

The statistics set forth by Professor Mitchell in a preceding paper† very carefully brought out the points that indicate that a marked amount of consideration must be given to the haulage problem to adapt it to local conditions as encountered. Professor Mitchell's paper showed a spread of from 10 tons per man to 50 tons per man on the production crews, which spread can be recognized as entirely too large when one considers that the product is being sold at a price within very close limits. While a portion of this spread may be explained by different methods in cost accounting as to what employees are applicable to the production crew, it cannot all be blamed on an accounting method. A good portion must be explained by the improper application of equipment to the needs incurred by the local conditions. It is in an endeavor to go into this phase that this paper is given.

It will be necessary to qualify the points covered in order to keep the subject from becoming too wide in scope. Along this line, conditions will be only those involved in bituminous coal mine practices and only the haulage of coal that is loaded with mobile loading machines in seams 4 ft thick and better.

Haulage Methods Employed

The Consolidated Coal Co. (Illinois) has been operating on a modernized basis for about 20 yr. By *modernized* we mean the loading of coal with mobile loading equipment. In review of haulage methods, it must be remembered that the main line electric haulage locomotive has been in service for some 40 or 50 years. To augment this main line haulage, the mule did a quite satisfactory job during hand-loading days, but was later replaced by the gathering locomotive, either battery or cable-reel operated. This type of haulage was the method in common use at the time the mechanical loading of coal became of major importance and the success or failure of loading machine installations may be largely accredited to haulage facilities in the early days; that is, in the period around 1930.

The Consolidated Coal Co. had its struggle with this problem beginning in about 1927 or 1928. We were somewhat fortunate in that we were already using gathering locomotives powered by storage batteries at the time we introduced mechanical loading. This meant that we did not have to spend as much time training motormen and trip riders as many mines did that utilized only mules for their secondary haulage. This was an advantage, and our efforts could be focused on ways and means of providing adequate car service to the loading machine.

Like most operations, we endeavored to gear the car service to the capacity of the machine, with a large crew endeavoring to load the maximum tonnage that the machine would handle. Toward that end our first move was to install switches

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* Vice President, Consolidated Coal Co. St. Louis, Missouri.

† D. R. Mitchell et al.: Shuttle-car Haulage. AIME *Coal Tech.* (May, 1947—TP 2198); this volume p. 149.

through each crosscut so that cars could be fed from the switch in the crosscut to its place behind the machine as the loaded car was being pulled out. This meant a maximum effort toward tracklaying, with the necessary timbering that would accompany a method as outlined, which meant that a high tonnage must of necessity be loaded to take care of the large working crew allocated to the machine. Several years of this brought us up with the realization that dispensing with the extensive tracklaying program would allow us to shorten the crew and the machine could load a smaller number of cars and still give the same unit cost.

Since that time our experience has convinced us that often it is more economical to install additional modern high-rated equipment, operating them with a shorter crew, and utilize more units of the equipment to produce the desired amount of product. This means that a delay on any one section would not involve as many men as though the equipment were being operated at near peak capacity. The short-crew method means that the reduction in men on the section tends to deplete the lower earning bracket and raise the average earnings per man; but this is offset by the fact that more tons per man employed may be secured. This increase in tons per man must be of sufficient magnitude to offset the interest on the investment and the increased depreciation and obsolescence. This figure will vary over a wide range, depending on the life of the property, the type of equipment, and the method of accounting presently employed.

The period up to approximately 1936 and 1937 was made up largely of endeavors to facilitate car changes, in order that the loading machine could be utilized in loading of coal as much as possible during the 7-hr period. The studies indicate that when a loading machine was operating about half the day, an operator could feel assured that a fair cost would result; but very few installations could boast that the loading machine was loading coal into the pit car 210 min out of the 7-hr day. The problem then resolved itself into devising some method of securing a higher output per man employed. This could be accomplished either by reducing the number of men or increasing the output with the same number of men, by installing additional equipment of the kind required. It was at about this time that the conveyor came into prominence as a method of haulage, to enable the coal operator to produce a larger "tons-per-man-employed." The conveyor was looked upon with some amount of skepticism as being something that would really be handy to have, but one about which there was some doubt as to its ultimate efficiency in increasing the tons per man employed.

CONVEYOR METHOD

The Consolidated Coal Co. made a thorough investigation of the conveyor method and decided that the best way to utilize a radical change of this kind was to start with a new mine and put the system into effect 100 pct. We started by sinking the slope with conveyors and installing an inclined conveyor for elevating the coal from the coal seam to the preparation plant. This conveyor was a success from the start and over a period of years has handled the hoisting problem in a practically trouble-free method at a greatly reduced cost, not only in manpower and maintenance but also in the actual power required to do the hoisting. Our figures indicate that over a period of years the power required for hoisting averages about $\frac{1}{10}$ kw-hr per ton per 100 ft elevation through which the coal is hoisted. This is about one half the power requirement of other methods of elevating the coal according to the best information we have been able to get, and our cost includes the power consumed on all three shifts, including the time the belt runs idle, is used to hoist rock, or for other purposes. Our experience indicates that our original estimated life of some 5 to 10 yr for the belt itself was approximately correct in that we are at the 10-yr life rather than at the minimum that we first considered.

The method of putting the coal onto the belt is just as much a part of the haulage as any other phase and our experience has indicated that the economical trouble-free method is to use a large hopper or surge bin that will act as a reservoir and enable the coal to be fed onto the hoist belt at a steady rate, which tends to produce a maximum efficiency in the tipple preparation. We have found that this bin is just as important when coal is brought to the bottom on a conveyor as when it is brought to the bottom in cars, inasmuch as it allows short-time surges to be accommodated without "slugging" the tipple.

It may appear that we are approaching the haulage question from the wrong end, in that we are first considering the final haulage rather than the service haulage. However, it is our contention that a method must be devised that will take care of the face haulage at its utmost peaks, and that if this is not done any advantages of new methods in face haulage will be handicapped and perhaps discarded when they really have merit.

Regarding main line haulage, the experience of our company indicates that large drop-bottom mine cars between some intermediate point and the hopper feeding the hoist belt is the final solution, providing of course that excessive grades, say over 4 pct, are not encountered on the route. At our Buckhorn mine we first fed the hopper by means of mother belts, grandmother belts, and similar equipment, and later replaced it with drop-bottom mine cars of some 10 or 12 tons capacity. Our figures indicated that this not only meant a smaller investment but also meant a reduction in personnel. They may sound contradictory; however, to ensure a proper continuous operation of a belt conveying system, it is necessary to have it properly patrolled and inspected continuously. This is necessary because of roof falls and coal spillage, which evidence themselves from the start and seem to occur at some of the most unexpected places. It can readily be seen that a fall on a main line belt would choke off all production, and it could not be restored again until the fall was removed and the belt line put back in service. If the belt has become cut or torn, it must be spliced or repaired, any bent pans must be replaced or straightened, and any bent rollers must also be replaced. Besides all this, a fall generally knocks the pan line out of alignment and it is necessary to re-align the pan line and adjust the tracking of the belt to prevent unnecessary spillage.

We recently started another modern mine and thought perhaps our information was erroneous, since our experience with belts on other than final main line haulage was practically without complaint. However, after about a year and a half with belt haulage all the way, we found that the figures from the new property checked with our original figures and we utilized the nonoperating time during the last contract negotiating period to replace the final belt haulage with large drop-bottom cars. Incidentally, at this time we increased the hopper from about 100 tons to about 350 tons, to accommodate surges on the bottom and delays on top. Our final hopper, which is in the process of construction, will have a capacity of 2000 tons, which will handle the product over a considerable period and should preclude "blowing out" on account of major breakdowns in the preparation plant.

Loading

As to the method of loading these large drop-bottom mine cars, we have found that this may be done very nicely from conveyor belts in the panel entries. In fact, we find that it is advantageous to discharge several panels onto a secondary-entry belt and take the load from this belt into the large mine cars for final haulage. This groups the belt installation in one locality, which lends itself readily toward being looked after by a single belt patrolman. The aforementioned method allows us to drive our main haulage track entries along a contour that makes for easy locomotive haulage with only secondary and panel entries belted from the territories where excessive grades would be encountered.

It must be realized that the method of transporting coal by belts into mine cars and then into a hopper cannot be arrived at blindly, but that the property must be drilled and contoured prior to the layout of the mining system. With regard to the system of mining in driving main entries, we have found that as cheap a method as any is to extend the main line track and drop a couple of cars toward the face to be loaded by shuttle cars, which in turn are loaded by loading machines at the face of the entries. The large mine cars are not end-loaded, as proper loading could not be handled in this way. Instead, the shuttle car discharges over the side, either by means of an elevating boom or by means of an unloading conveyor into the large car that is placed by a battery locomotive brought up to the face for that purpose. If the track is in entry No. 1 and a fall is being loaded in entry No. 1, the shuttle car will be loaded at the face and will then proceed through the crosscut to entry No. 2 and then along No. 2 to the next crosscut, where it will again cross back and load the large drop-bottom car, which is pushed up in place on the No. 1 entry. This may seem like an extra amount of transportation, but our experience indicates that it is more satisfactory and economical than trying to end-load in the track entry.

SECONDARY HAULAGE

This brings us up to the secondary haulage, which is in fact the critical phase in the economical mining of coal. It is upon this secondary haulage that the success or failure of a modern mine depends. There are many things to be considered in designing this haulage that may make one type successful in one field and unsuccessful in another. The Consolidated Coal Co. has utilized practically every available type of secondary haulage except shaker conveyors, which were not believed to have sufficient capacity to handle coal from our mobile loading machine.

Probably our most economical operation was the use of chain-flight conveyors loading onto belts and the territory manned by a very short crew. Utilizing this method, we were able to work out four rooms more than 300 ft long per week, driving the rooms an average 26 ft wide in coal averaging 7½ to 8 ft high. Sufficient crosscuts were driven to procure an extraction approximating 65 pct; however, this was some 10 yr ago and since that time we have had to modify our methods to secure compliance with recently passed legislation. Legislation in Illinois provides for not less than two men on a loading machine, not less than two men on a cutting machine and not less than two men on a drill. This of necessity disqualified us on the higher wage scale gang-work where we utilized a five-man crew mining four rooms with the conveyor system, the coal being broken down "on-shift."

In an endeavor to reestablish our productivity, we adopted shuttle cars as face haulage and endeavored to run the territories at an increased output to offset the additional men employed. The shuttle car has proved itself equal to the occasion and we have found that a battery-type shuttle car can operate efficiently on grades up to 6 pct, beyond which we find that it is economical to utilize a cable-reel shuttle car, which we work on grades up to 8 or 10 pct. The shuttle car has the flexibility that enables one to haul around grades caused by pronounced local rolls, which are prevalent in the Illinois No. 6 seam. When a sharp short grade is encountered, haulage is then rerouted to pick up a place from the adjacent place through a crosscut, perhaps doubling the haulage, but at the same time halving the maximum grade. The flexibility of the shuttle car permits the working out of rooms where hazardous top conditions are encountered, by hauling through adjacent rooms and crosscuts.

We have in operation at the present time several methods of face haulage in an endeavor to secure information as to which would be the best to adopt as standard. In territories where few extreme grades are encountered, we have found that one of the most productive methods is to drive the rooms parallel to the entries with key crosscuts short every hundred feet or less, heading off the rooms to shorten the haulage to the panel entry. While this necessitates a considerable number of permanent seals, it does produce a productive section when using five rooms parallel to the entry, the faces being equally advanced, the coal cut with a mobile cutting machine and loaded with a mobile loading machine served by two shuttle cars. It is in this type of installation that the battery car really works to an advantage, in that there is no crossing of cables to worry about and the coal from all five rooms is loaded at a common point on the panel entry, and an unloading conveyor at this point serves two or more shuttle cars. In handling the coal between the face haulage and the slope bottom on this system, we have found that a trip of large drop-bottom cars can be placed on the panel entry adjacent to the rooms and thus eliminate the belts on the panel entries. This allows an extremely flexible layout in that the large drop-bottom cars are handled in trips of five, which provides sufficient capacity for the loading out of one room. The cars are not uncoupled but left in groups of five, whether there be one or two empty cars in the trip or not.

This method is still in the experimental stage and we have much to ascertain as to what might occur at a later date on the mined-out territory. On our present section, we have not run our rooms beyond 3000 ft long as we were a little apprehensive as to what might occur after a period of time. The system, however, does yield a very nice productivity per man employed with a gang working in the development of the entries and a crew working the rooms parallel to the entry and on one side. It is our intention to retreat with the rooms at 90° to the entry, putting two crews on the panel entry, one retreating from the face and the other retreating from a point about midway in. This will extract the coal in about half the time it took to advance and the inmost territory will be protected by the solid block remaining between the middle point and the point of operation. As mentioned before, this method is in an experimental stage and we do not have much data on the system at present.

Earlier in this paper it was mentioned that the main haulage entries were laid out on the predetermined contours to keep the final haulage grades at a minimum. From this main line haulage we experience prevailing grades of from 2 to 10 pct, which, with the ordinary pit-car method, would present quite a problem. We have laid out the extraction to provide belt haulage on the secondary entries in the part where the grades prevail at 8 or

10 pct. Off these entries the panel entries are equipped with belts to feed the secondary entry belts, haulage between the panel entry belt and the face being accomplished by shuttle cars. In extreme cases the shuttle cars have to be utilized in hauling with the grade, discharging on the belt, which belt in turn discharges on the belt in the secondary entry and this belt elevates the coal to the main entries and into the large drop-bottom mine cars. These heavy prevailing grades preclude the working of rooms on both sides of the panel entry, which doubles the amount of development work necessary to extract the coal in this area.

By the combination of shuttle cars, belts and large drop-bottom mine cars, we are now able to extract coal that 10 yr ago could not be mined because of the excessive cost at a mine operating in a highly competitive field. We find that with this combination we are now able to extract the coal, and while the productivity per man is not as high as on more nearly level runs, we find that we are able to operate in competition with surrounding properties that do not have the extreme conditions. It might be noted that with a belt in the panel entry, each room neck is a potential loading point, which lends itself ideally to shuttle cars equipped with an elevating discharge boom.

Results in an Experimental Mine

The results of our efforts in haulage during the past 10 yr have been so encouraging that we ran an experimental mine operating in 4 ft of coal immediately below the No. 6 seam which we had previously extracted. We ran the experiment at this operation for over a year and a half, being handicapped by labor that was accustomed to work in high coal and that could not readily adapt itself to the 4-ft seam. We found that we were able to produce coal in this 4-ft seam in competition with the high coal, but that we were just about breaking even, since we were unable to secure the services of low-vein coal miners, because of the manpower shortage during the war period. In spite of this handicap, we did operate by using low-vein loading machines, mobile cutting machines, and low-vein shuttle cars loading onto belt conveyors in the panel entries, which in turn discharged onto a belt at the main entry, this main entry belt discharging on an inclined conveyor elevating the coal to the large mine car that was in the No. 6 seam approximately 40 ft above, for transportation to the hopper already located in the No. 6 seam.

The transportation of supplies and material for mining was a major problem which was overcome by building "material jeeps," storage-battery powered, the highest point of which was the top of the tire, a distance of 28 in. above the mine floor. This allowed a flat carrying area between the wheels for the transportation of material, and further material was carried on a trailer pulled by the low-vein "material jeep." We have just this month finished our experimental extraction in the 4-ft coal and know that at such time as may be deemed fit we can extract this 4-ft coal at a cost somewhat greater than the No. 6 coal. This time, however, will be at a later date, when the diminishing field in Illinois will provide a more advantageous market than the present period.

Summary

By utilizing a combination of shuttle cars, belts and large pit cars, we can extract acreage that otherwise would be left intact, and the saving resulting from the use of this combination of equipment will more than offset the increased capital investment, with its accompanying increase in maintenance, depreciation and obsolescence. The question of the correct combination of equipment is one that must be determined by the operator to meet the conditions to be encountered; however, these conditions can pretty well be established by drilling and contouring to secure an advanced picture of the haulage requirements to extract the coal in a given field.

In turning back to the "final haulage" with large mine cars rather than belts, there are a couple of points that may be of interest. Our information over a period of years indicates that the power consumption with the locomotive is approximately the same as the power consumption with the belt "final haulage," which is in sharp contrast to the experience in the hoist belt versus skip hoisting. This no doubt results because the belts run continuously when utilized as a "final haulage" below ground while the locomotives run only periodically. However, the demand load is increased sharply on locomotive haulage versus the belt when the locomotives get together on the acceleration cycle of their trip. This increased power cost, however, is more than offset by the increased cost in initial investment and the decreased cost in manpower when compared to belt "final haulage."

In our final haulage, we do not handle the cars separately at any time, but handle them in groups of five, which means that operation is accomplished with a minimum number of cars, each car completing a cycle approximately every hour or less. In running the short car trips, transportation that makes for a flexible operation is relatively fast and one in which trips can be run on a priority schedule to territories that hit fast loading conditions, whenever the opportunity presents itself. This of course resolves itself into a dispatching system, which we have found pays for itself in taking the coal away from a territory as fast as it can be produced when conditions are encountered that tend toward a period of peak productivity.

Our recent requirements necessitate that main line haulage locomotives operating on fast schedule be equipped with what might be classed as "modern gadgets." We use air brakes, dynamic braking, air sanding and even a final refinement in specifications by the inclusion of rail brakes, similar to those used on modern street cars, which are to be automatically brought in to operation at a point where the dynamic braking begins to become ineffective. This is to ensure a smooth operation on the deceleration cycle with a minimum of wear on equipment and a minimum of effort on the part of the motormen. While the air brakes in this arrangement might appear to be superfluous, we consider their presence essential as an emergency precaution in the event of failure in the dynamic rail-braking program.

The results of our study during the past several years have indicated that the advanced design in locomotives makes it essential that we replace our present locomotive equipment with modern equipment just as surely as we were forced to replace our obsolete face equipment with modern equipment to operate competitively with strip-mine operations in our own field.

The Foundation of Safety Engineering and Planning

By J. D. Cooner,* Member AIME

(New York Meeting, February 1948)

Since my working life of 32 yr has been spent in and about the anthracite mines of the Hudson Coal Co., and the previous 4 yr in a college school of mines, I can write best about the safety program of a coal mining company. However, most of the suggestions, statements and practices mentioned herein are applicable to the members of a safety organization and program of almost any industry. It is realized that industrial officials actually interested in real safety work are fully acquainted with the requirements of a practical safety program. Therefore, for such persons there may be nothing new in this paper.

The Hudson Coal Co. of Scranton, Pa., employs 7000 men and produces 5,000,000 tons of anthracite annually at nine collieries scattered for a distance of 40 miles throughout the Lackawanna and Wyoming valleys of northeastern Pennsylvania. For the past 17 yr of the above mentioned 32, I have been engaged in safety work as a safety inspector, assistant mine superintendent, and at present the safety engineer. I have also had the privilege of visiting coal and metal mines in other parts of the United States and Canada.

We Americans are the most careless people in the world. If any one doubts this statement why are 100,000 of our people killed accidentally annually? Or why during the almost four years we were engaged in World War II were more of our people killed accidentally at home than were killed in our army, navy, marine corps, coast guard, and other services during the same period? We are careless everywhere—in our homes, on the highway, at work and at play. The purpose of this paper is to outline the requisites of a safety department and its program to stop the accidents occurring to employees at work in mines or industrial plants.

The first requisite for a safety program is that the management must be convinced of the need and value of a real safety program. By "management" is meant each department head, from the president down. Otherwise a good safety department with a good safety program will not succeed because it will not have the proper personnel and equipment to work with, its rules and regulations will not be enforced, and the members of the safety department will be belittled in the eyes of most of the workmen and supervisors.

Next, a safety engineer and his inspectors should have worked in the production end of the mines as workmen and also in an official capacity. They are then in a position to have practical and first hand knowledge of the accident hazards of the industry, how to prevent them, and in general the problems of both the workmen and officials.

Some men are appointed as safety inspectors when they are physically unable to carry on as mine officials. Such men may know the safety end of the industry and even be safety-conscious, but if they are physically unable to be mine officials they are also physically unable to be safety inspectors, as a safety inspector in our mines must walk from 2 to 4 miles on each daily inspection and visit each working

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face in the section whether the coal be 20 in. or 20 ft in height. Also, a safety inspector's job should be a stepping stone to a better job such as foreman, assistant superintendent, or safety engineer. Such a practice will be an incentive for a safety inspector to do a better job. Therefore, the safety department should not be a place for pensioners.

A good safety engineer need not be a superman. He should, however, have the same qualities that are needed by any other successful supervisor who has problems and people with which to deal. Above all he must be a firm believer in safety and at all times practice what he preaches. The average person on or off the job quickly observes whether or not a safety man is safety-conscious. A firm believer in safety will automatically do things the safe and proper way. He will park his automobile at the proper place and get out of and into it the safe way. He will wear all the prescribed safety clothing and other safety articles on the job and above all will follow the safety rules and regulations. He will remove safety hazards on the job and ask, not tell, the foreman or others to do likewise. Besides *acting* safety he will *talk* it whenever the opportunity presents itself. Such a safety man is bound to spread the gospel of safety.

No workmen or supervisors go looking for safety inspectors. Since inspections are a necessary part of a safety program they should be made as pleasant as possible. In our mines the safety inspector generally goes with the sectional forman and visits each working place. This takes from 4 to 6 hr. First, so as not to delay the sectional foreman, the safety inspector should be on the job before 7 A. M., the time the sectional foreman starts on his daily visit. When safety hazards are found, the inspector should allow the sectional foreman to do the talking to the workmen about correcting the hazard and the administering of a reprimand or summary discipline when necessary. In fact this is a "must" with our company as the sectional foreman is the boss in the section. If and when the sectional foreman does not mention a hazard that the inspector has noted, the mere pointing of a finger at the hazard or a few words from the inspector are sufficient. During the inspection the inspector should mention recent accidents and particularly any that have occurred in that section or have been caused by any hazards noted in the section. At the end of the inspection he should advise the sectional foreman of the criticisms his report will contain. If the inspector has noted anything good or unusual in the section he should compliment the sectional foreman and do likewise to any workman whose working place is in good condition. If an inspector is tactful and friendly he is paving the way for workmen and supervisors to feel that he comes to help and not to criticize. At the same time an inspector must be a keen observer and report all hazards noted. Otherwise, even those being checked will consider him a poor inspector and have little regard for him as a safety inspector.

It is a well known fact that no matter what a group of people is working for there is generally a good chance for harmony and success when fairness and cooperation are present. The safety engineer surely must have these qualities. He can be fair and still be a good safety engineer but he should not be a "yes" man. When investigating accidents he can be fair by basing his conclusions on facts and evidence observed at the scene of the accident and the testimony of any witnesses including the supervisor in direct charge. When the investigation is completed the safety engineer should discuss his conclusions with the colliery superintendent. If their conclusions are not the same they should thoroughly discuss the facts but not write letters. If after the discussion they cannot agree the safety engineer should send through his report on which the colliery superintendent may state his exceptions and reasons therefore. Such controversies with us do not occur often, but when they do occur and the general manager makes the final decision, the safety engineer should forget about the decision whether it be for or against him. The real purpose of an accident investigation should be to ascertain the cause of it so that the facts may be passed on to all supervisors who in turn should pass them on to their workmen so that similar accidents may be avoided.

There are numerous ways in which the safety department members may cooperate with the officials of the mines and other departments. Some of those ways have already been mentioned because being tactful, friendly and fair are means of cooperation. They should always secure safety articles and information for the mine supervisors when requested. When the promise is made to furnish such items that promise should be fulfilled. The safety department can and should be eager to help keep the colliery gas detecting instruments, anemometer, and such, in good repair and see that such items as air sample bottles and smoke tubes are provided. The safety engineer should attend all colliery meetings, day or night, when asked by the colliery superintendent and be ready to talk at such meetings. If not asked to talk he should not resent it, but above all he should be prepared if called upon to spread the gospel of safety.

Another means of cooperation is for a safety engineer to have an open mind for safety suggestions from anyone and should welcome constructive criticism. After due consideration the person making the suggestion or criticism should be given a definite answer and whenever practical these suggestions should be given a trial.

A safety engineer should be above the average writer and speaker so that he can put safety facts on paper or talk to a group in an understanding and convincing manner. These facts whether written or oral should be in the language of those who will read or hear them. It is realized that only a few persons are good writers and talkers but the safety engineer who does not have these abilities will be handicapped and therefore should spend considerable time in developing them.

A safety engineer has many trials and tribulations. At times it seems that the more safety effort that is put forth the more accidents there are. At such and other similar occurrences he must redouble his efforts. If he becomes discouraged and proceeds accordingly perhaps both he and his program may be lost.

Safety engineering is the planning and execution of those rules, regulations and practices necessary for the health and safety of persons employed in and about industry and for the protection and preservation of the property connected therewith. Most safety programs in any industry have certain common practices which are to inspect, investigate, analyze accident statistics, keep the management informed through the proper presentation of these statistics, watch out for so-called safety incentives that are outdated and substitute new ones, and in general advertise safety continually throughout the property. These various practices must be made to fit the individual plant or company.

The questions of inspections and investigations have already been mentioned. For inspections to be effective the report covering same should be sent through at once. Our inspectors make out their reports on the same day the inspection is made. They state the safety hazards found and their locations. The original and one copy of the report are left with the mine foreman and the second copy is sent to the safety department. There is a column for the sectional foreman to state his comments or action taken. The original is then signed by him and the mine foreman and sent to the colliery superintendent for his information. If the hazards are numerous or serious the superintendent discusses them with the sectional foreman and mine foreman before he adds his comments and forwards same to the safety engineer who reviews it and sends it to the general manager with any necessary comments.

We have periodic inspections by state and federal inspectors. It is the definite policy of our safety department and mine officials to aid and cooperate with these outside inspectors. The opinions of others regarding safety work are nearly always beneficial.

The available accident statistics should be sufficient to show the trend of the general safety performance and principal contributing factors and in turn used to advantage. We have and use the following statistics:

1. Statement showing number of eye injuries and lost time injuries for the current month and total lost time injuries to date for each sectional foreman, mine foreman, outside foreman, and total colliery. These statistics are published in the company's monthly safety magazine, *The Safety Commentator*, which is sent to each official and most of the employees.

2. Compensation department statement showing the number of fatal, serious and minor injuries by causes for each colliery monthly and yearly to date. My monthly report to general manager contains a summary of this statement with my comments.

3. A safety rating statement is made up monthly showing for each supervisor the number of lost time injuries, man days lost, man hours exposure, with the frequency and severity figures for each one. A quarterly summary is made from these figures and sent to all supervisors and department heads.

4. Safety department issues monthly statement showing fatalities by collieries yearly to date with monthly totals compared to previous years. This statement is posted on colliery bulletin boards and published in *The Safety Commentator*.

5. Monthly statement showing tons of coal mined per fatality for each colliery and total property.

6. Statement showing the number of lost time injuries and number of cases of discipline monthly and yearly to date for each colliery official.

7. Annual statement from the medical department showing the number of injuries by classification such as fractures, lacerations, and others.

8. The Dept. of Mines, Commonwealth of Pa., issues each two months *The Safety Sentinel*, which gives the details of each mine fatality, tons per fatality, and other details, for each mine and company.

With such information available, statistics can be worked up in almost any form for the information of the safety department and management. No accident statistics are of any value if not used and not accurate. They should not be altered to make a poor record look good. A safety engineer who attempts to do this deceives no one but himself.

We endeavor to keep safety continually before our workmen and colliery officials by publishing or posting the statements of statistics mentioned in (1), (2), (3), and (4), and by

a. Awarding a large well-made safety flag quarterly to the mine foreman having the best safety record at his mine as per the safety rating figures mentioned in (3) above.

b. Sending five mine officials annually to the National Safety Congress for having the best safety records annually in their respective areas.

c. Having an organization known as the Safety Key Men in which each supervisor becomes a member when there have been no lost time injuries to his men for a minimum period of twelve consecutive months.

d. Awarding cash bonuses quarterly on a sliding scale to sectional foremen who keep their lost time injuries to a minimum.

e. Publishing monthly magazine known as *The Safety Commentator* and sending it to the homes of our employees.

f. Participating in the National Safety Competition annually. Our mines have won the anthracite bronze trophy three times and also received numerous honorable mentions.

g. Making first aid available to all employees and having annual first aid meets.

h. Having outside and inside bulletin boards. Safety bulletins must be practical, attractive and applicable to your industry.

i. Presenting quarterly safe worker buttons for all employees working in sections where there have been no lost time injuries.

j. Having supervisors wear white safety hats from the first of each month until he has had a lost time injury in his section.

We have tried safety meetings for all employees where we had good entertainment, valuable door and safety prizes, and short safety talks, but the attendance was too small to justify the expense. Safety certificates were issued quarterly to supervisors having no lost time injuries in their sections for 3, 6, 9 and 12 months. After several years so many were awarded that they lost their value and were discontinued. Our *Safety Commentator* would give the facts in connection with each fatality on our property. Some of the officials considered that this information in the homes of our employees had adverse influence upon some mothers who would not allow their sons to work in the mines and this information was therefore discontinued.

The above mentioned items are examples of safety incentives that were discontinued for the reasons given. A safety engineer and other officials must continually be on the lookout for the safety incentives that are growing stale and new ones to take their places. Such new ideas are learned from 1. Reading mining, industrial and safety magazines. 2. Visiting mines of other companies and extending the same privilege to other safety men. 3. Belonging to organizations such as the AIME, National Safety Congress, Mine Inspectors' Institute of America, and taking part in their programs. 4. Having an active association with the safety men from all the companies of your industry in your community. This is important.

With all these methods of attempting to make the workmen and supervisory officials safety-conscious, there remain a certain number of workmen and a few supervisors who show little response to the safety program. It is this group that presents a real problem to an organization that has a well organized safety program. Such officials should be dismissed from supervisory work if they do not respond to the advice and warnings of their superior officers. These particular workmen should be well known by an alert supervisor whose duty it is to pay special daily attention to them by instructing and insisting that they work safely. If such workmen do not then respond they should be disciplined when found disobeying instructions or safety rules. You must touch some workers' pocketbooks to make them safety-conscious. Discipline must be just and given in the proper manner.

Unions could be of much help to the safety programs in their respective industries. Many of the U.M.W. of A. officials in our field are interested in the safety movement but they have no known safety program. It is my opinion that many of the workmen who now pay little attention to the employer's safety program could be reached if the union would cooperate by requesting the local unions through a safety director of their own to hold safety meetings and otherwise urge the men to work safely. Such a safety program has recently been started in District 29 of the U.M.W. of A. at Beckley, W. Va., where there is a district union safety director who is working with the mining companies. This cooperation has already shown good results.

If there were a fire or explosion in your mine, or an explosion in your industrial plant, or if a heavy snow or wind storm caved in your plant roof and trapped workmen, including perhaps you, would you have to wait for outside help? Therefore, each safety program should include the training of enough men to take care of such emergencies. There should be on hand the proper type of rescue apparatus and emergency supplies for emergency purposes only. On such occasions immediate help is the most effective.

A safety department should not be the dumping ground for those many odd jobs that do not belong definitely to any department, as time spent by the safety engineer on other work means just that much less time for safety work. When other work is assigned to the safety department perhaps there is another opportunity to advertise safety. In my case I am the contact man with a local high school where vocational mining is being taught to the boys. I see to it that these boys hear a lot about safety. I also have charge of the company students who are being trained to become supervisors. Their ten months' schedules include two months in the safety department.

A safety engineer and his department should report directly to the operating head so that he will be under no obligation to any mine or plant official. He and his inspectors must report the facts and conditions as they find them. In safety when facts are hidden they generally turn up as injuries.

A properly planned and executed safety program will pay for itself for it will prevent human suffering of body and mind of both the worker and his family and at the same time save the employer money in medical care, compensation and production. It is my sincere belief that nearly all accidents are man made and can be avoided.

Safety Considerations for Flame-resisting Trailing Cables

By F. E. Griffith*

(New York Meeting, March 1947)

Several disastrous and a great number of near-disastrous mine fires have been started by igniting the combustible conductor insulation and outer covering of trailing cables. Those who have had experience in fighting such fires know that they are extremely difficult to combat and extinguish owing to the irritating and noxious gases, fumes and vapors evolved from evaporation, distillation and combustion of the sundry compounds used in the construction of cables.

When electric-motor-driven coal-cutting machines were first introduced in coal mines, the electricity was transmitted to the apparatus by two single-conductor trailing cables. Later, two-conductor parallel and concentric types were developed. These early cables were made with the materials available at that time with relatively little thought given to using new materials or utilizing the old to the best advantage. The copper conductors were stranded to give flexibility, but there were too few strands and the method of laying them was such that they did not endure well the stresses of twisting and pulling, to which trailing cables used in mines are usually subjected. The rubber insulating compounds used were not satisfactory in resisting water, high temperature, and the crushing stresses encountered in mine service. The outer coverings were cotton or jute braids saturated with tars or asphalts having relatively no abrasive resistance. In fact, the only consideration given to trailing cable design because of its extremely short life, was to construct it at the lowest possible cost.

With increased electrification and mechanization, more serious problems were encountered. Cables used in mines were exposed to more severe conditions than in any other industry. The excessive overload currents, poor heat dissipation, severe abrasion, crushing loads from falling material, runovers, and any of the other things that tend to destroy cable, called for vast improvement in trailing cable design for mine use.

About 1918, one manufacturer who recognized the need for improving the toughness of cables in order to obtain greater safety and longer life, developed a special rope-stranded conductor, a superior rubber insulation compound and a tough rubber sheath or jacket compound. As a result of this type of construction, broken conductor strands were practically eliminated, the insulating material withstood high temperature without undue disintegration, and the jacket, similar to that used for automobile tire tread, had high abrasion resistance, high tear resistance, and was waterproof. While the initial cost of these cables was higher than the jute-filled braided types, this disadvantage was more than offset by increased safety and longer service.

Cables with these improved characteristics were first installed in mines in 1920. Since 1930, the use of braided trailing cables in coal mines has been almost completely supplanted by the more superior types, employing high-grade rubber compounds for insulation and sheath.

The next steps taken to improve the

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toughness of trailing cables were brought about when the United States Bureau of Mines developed a crushing test. Cables that passed this test were especially recommended by the Bureau for use on mining machines.

This test consisted generally of running a car loaded with pig iron having a total weight of 7 tons, over the cable specimen. If a failure of the electric-current-loaded conductors did not occur with a frequency greater than one in ten tests, the cable was recommended. Making cables to pass these tests involved changes in construction, many of which enhanced the overall safety and gave added life to the cable.

Following the outbreak of war in 1941, the natural rubber used in the construction of trailing cables became unobtainable. This compelled the cable manufacturers, after most of the supply of reclaimed rubber had been exhausted, to use synthetic rubber which required some changes in techniques. The cable manufacturers accepted the challenge and, as a result, developed cables with characteristics equal to those obtained with natural rubber. They learned or are learning that the proper compounding of neoprene (a type of synthetic rubber or chloraprene) gave highly desirable flame-resistant properties to trailing cables.

Flame-resistant Cables Recommended

At a meeting of the Coal Mining Institute of America in December 1944, at Pittsburgh, Pa., a prominent coal-mine operator praised the safety value of the flame-resistant properties of trailing cables used in his mine. He also proposed a motion to the effect that the Institute go on record recommending that trailing cables, having these properties, be used in all coal mines, and, further, that the Institute's secretary write all coal-mine department administrative heads of the various states advising them of this action and suggesting that they endeavor to have their legislatures enact laws requiring their use.

The Honorable Richard Maize, Secretary of the Pennsylvania Department of Mines, took immediate action. After a preliminary investigation, he prepared a suggested bill which was presented to the General Assembly. It was enacted by the General Assembly as Act No. 206 and was signed by the Governor of the Commonwealth of Pennsylvania on May 15, 1945. The Act, in part, reads as follows:

"... relating to the use of trailing cables on portable electric machinery in coal mines ... it shall be unlawful to operate any portable electric machine by means of a trailing cable ... one year following its final passage ... unless such cable is safely and efficiently insulated by flame-resisting material and approved by the Secretary of Mines ..."

Investigation

The Secretary, thus authorized, directed an investigation of the flame-resistant properties and characteristics of cables that would be suitable in these respects for mine service. The flame resistance cable requirements of the Navy and Underwriters were studied. While cables passing these test procedures were excellent for the conditions under which they were to be used, these test procedures did not take into consideration such factors as internal heating from overload currents which often occurs in mines. It was the Secretary's thought that the cable specimens should be tested in a manner that would simulate the most severe conditions they would be subjected to in coal mining.

It was also his thought that the State's requirements for flame-resistant properties of trailing cable should not be so stringent as to preclude its manufacture. In working out the requirements and specifications, it was necessary to develop a testing procedure by which cable samples submitted by the various manufacturers could be tested and the test data carefully analyzed to find the best performance in flame-resistant properties. This test would ascertain the cable showing the best characteristics.

The Pittsburgh Coal Company graciously offered and provided the testing facilities. For their cooperation and helpful assistance, the Secretary of Mines and his representatives are deeply grateful.

It was thought that a relatively long (6 ft, as compared to about 15 in. used in other tests) length of a cable sample should be used to prepare each test specimen, the reason being that in order to simulate a coiled cable or one that was wound on a reel, the cable specimen should be lapped back on itself and firmly held in this position. Also, since cables as used in mines are ordinarily lying flat on the floor, or suspended almost horizontally, the progress the flame made while in this position should be the principal factor in determining the cable's value in preventing flame propagation.

In the first tests made, the samples of cable, all of which were understood to have passed the flame-resistant test of the Navy and Underwriters, were ignited with an acetylene-air torch with no preheating by overload current on the conductors. In each of these cases the flame was "self-extinguished" quickly and very little of the sheath and insulating materials was consumed. With subsequent tests on samples of the same cables, the cable specimen was preheated prior to ignition by imposing an electric current load on the conductors and waiting until the superficial sheath temperature reached a pre-set value. After this, the cable specimens were ignited and, for the most part, showed decidedly poor flame-resisting performance. With many, the flames reached a height of several feet and were so intense that all combustible material of the sheath and insulating compounds was consumed within a few minutes.

Many preliminary tests were conducted and much discussion with interested parties followed each series of preliminary tests. Finally, a tentative draft of the "Commonwealth of Pennsylvania, Department of Mines, Requirements for Testing the Flame-Resistant Characteristics of Trailing Cables" was prepared.

On February 21, 1946, Mr. Maize called a meeting to discuss the proposed requirements and the test procedure that was involved. Representatives from the Insulated Power Cable Engineers Association, Pennsylvania coal-mine operators, Bureau of Mines, and the Pennsylvania Department of Mines attended the meeting. The proposed requirements were accepted, in principle, unanimously.

Requirements for Testing

The statements contained in the Commonwealth of Pennsylvania, Department of Mines "Requirements for Testing the Flame-Resistant Characteristics of Trailing Cables" under the heading of "General" are as follows:

"These requirements cover the test procedure and test facilities that shall be used to determine the flame-resistant properties of cables. When a trailing cable is constructed in accordance with the requirements herein set forth, it shall be approved by the Secretary of the Department of Mines for use in underground coal mines of the Commonwealth of Pennsylvania.

"The requirements are based upon records of tests and coal mine experiences and the Secretary of the Department of Mines reserves the right to revise them as further experience and investigation may show to be necessary or desirable. He may withdraw his approval if the flame-resistant properties are not maintained to equal that of the specimen originally approved.

"For purposes of definition any cable

that is pulled or reeled shall be considered a trailing cable.

"Trailing cables which comply with the requirements will not be acceptable if they create other hazards not now found in present day cables.

"The cable manufacturer shall be required to provide the test facilities and conduct the initial tests. Tests on samples collected in the field will be conducted at U. S. Bureau of Mines.

"A manufacturer desiring an approval of his product shall make his request to the Secretary of the Department of Mines, advising him that he is prepared to conduct the tests in accordance with the requirements. Upon receiving the request the Secretary will send a representative of the Department to the place designated by the manufacturer. The representative shall observe and collect data, report the results of the tests and make appropriate recommendations to the Secretary as to whether the cable or cables tested meet the requirements and specifications as hereafter listed.

"A trailing cable that has met the minimum requirements and is approved by the Secretary of Mines, shall receive an approval number. For identification purposes the sheath or jacket of the approved cable shall bear an approval number which shall be prefixed by the letter P, and the conductor size shall be also shown. This mark shall be placed along the sheath covering at intervals not exceeding twelve (12) feet."

These statements outline the conditions under which cables shall be approved by the Secretary or under which he shall withdraw his approval; they contain a definition of a trailing cable; they state who shall conduct the tests; the procedure to be followed to obtain approval; and how the cable shall be marked for identification purposes. I do not believe they need further amplification or clarification.

Specimens, Facilities, and Test Procedure

Under the main heading "Facilities and Test Procedure for Determining the Flame-Resistant Characteristics of 600-Volt Trailing Cables," subheading "Specimen," the requirements contain the following:

Specimen

1. The test specimen shall consist of a six-foot length of cable and it shall be bent at its center to form a closed "U."

2. The sheath or jacket shall be removed from the terminal ends of the specimen a distance of approximately five (5) inches and the insulation compound shall be removed a distance of approximately two-and-one-half (2½) inches from the end of each conductor.

3. The bent cable specimen shall be clamped with three one-half (½) inch wide and 0.015 thickness gauge metal bands. The center of the first band shall be placed seven (7) inches from the terminal ends of the cable specimen, and the second and third bands shall be spaced eleven inches apart respectively, beginning the measurement from the center of the first band. See Fig 1.

4. The conductors of the cable specimen, except those to be connected to the source of power, shall be connected so as to provide a series circuit through the conductors.

These numbered statements give the length of the specimen and how it shall be prepared and connected to the source of power.

The reasons for being so specific with the dimensions and general arrangement of the specimen were to minimize the influence of variables that may change the test results.

Under the same main heading are two other subheadings: *Test Apparatus and Facilities*, and *Test Procedure*. The statements following each of these are self-explanatory, and it is believed need no elaboration. They contain the following:

“*Test Apparatus and Facilities*

“1. The source of electric current for loading the cable specimen may be either alternating current or direct current, shall have sufficient current capacity, and shall be provided with a means for closely regulating the electric current.

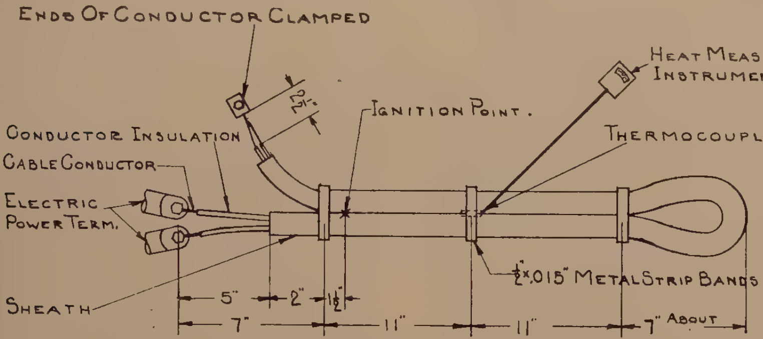

FIG 1—DETAILS OF SPECIMEN PIECE OF CABLE 6 FT IN LENGTH FOLDED PARALLEL.

“2. An accurate ammeter shall be provided for measuring the electric current load imposed on the cable specimen conductors.

“3. A suitable and accurate temperature measuring device or devices shall be provided to determine the sheath or jacket temperature.

“4. A rack shall be provided for supporting the cable specimen, and it shall have four metal rods spaced on 8-inch centers. The rods shall be wrapped with asbestos tape to minimize cooling effect, and the height of the rack shall be sufficient to permit the inner cone tip of a Tirrill (bunsen) burner, having a nominal bore of ⅜ inch, to contact the sheath of the cable specimen.

“5. A timing device shall be provided, such as a stop clock or watch equipped with a second hand that makes one revolution per minute and it shall have a minute indicator.

“6. A standard Tirrill (bunsen) burner for igniting the cable specimen at the hereafter designated time shall be provided.

“7. A room, hood, or canopy that is generally free from air currents shall be provided for conducting the tests.

“*Test Procedure*

“1. The cable specimen shall be placed in the rack and connected to the electric current source. The specimen shall be adjusted in the rack so that the band on the terminal end will clear the inner side of the first supporting arm by one-half inch.

“2. The current loading of the specimen conductors shall be 400 pct in excess of its normal current rating. The normal current rating for the cable conductor size under test shall be as follows:

AMERICAN WIRE GAUGE	NORMAL CURRENT RATING
14	18
12	23
10	31
8	41
6	54
5	63
4	72
3	83
2	96
1	110
0	127
00	145
000	166
0000	193

“3. The current load shall be maintained on the cable specimen until the sheath temperature reaches 350 degrees F and plus one minute of time has elapsed for ignition.

“4. The thermocouple of the temperature measuring device shall be placed under the center metal band at points of tangency between the sides of the cable specimen.

"5. The ignition of the cable specimen shall be by a Tirrill burner having an inner cone of three (3) inches in length. The tip of the inner cone shall be placed against the bottom sheathes of the folded specimen at a point one-and-one-half (1½) inches from the first metal band (terminal end) and allowed to remain in this position for one minute before it is removed.

"6. The heating time including one minute ignition time and the burning time (after the one minute ignition time) shall be noted and recorded. The total length of the charred or otherwise destroyed sheath shall be measured and recorded as inches propagation. The measurement of flame propagation shall be between the two extreme burned points on the folded cable specimen."

The "meat" of the "Requirements" is listed in the concluding paragraph under the main heading, "The Approval of a Cable," and is as follows:

The Approval of a Cable

"A cable will be approved by the Secretary of the Department of Mines of Pennsylvania, when a sample of it has been tested in accordance with and meets the foregoing requirements, and when the *propagation does not exceed fourteen* (14) *inches.*"

In other words, if the measurement between the two extreme burned points on the folded cable specimen exceeds 14 in., that cable will not be approved even though all the other requirements have been met. One year has elapsed since the Pennsylvania Department of Mines prepared and promulgated the foregoing requirements and specifications on flame-resistant trailing cables. A number of coal-mining companies are 100 pct equipped with approved cables, and other companies in Pennsylvania are rapidly approaching 100 pct installation. It may be of interest to note that coal and metal mining companies in other states are demanding that newly purchased trailing cables meet the flame-resistant requirements of the Pennsylvania Department of Mines.

At previous discussions of this subject several interesting questions were asked concerning the properties of flame-resisting cables that meet these requirements. These were:

Question: Has the flame-resistant trailing cable been tested under actual mining conditions? If so, for how long a period and to what type of equipment was it applied?

Answer: At least one manufacturer's trailing cables having these flame-resisting properties have been in service for a period of more than two years, and were used on all types of mine electric-motor-driven equipment.

Question: Does the flame-resistant cable compare favorably with other cables when used on cable-reel equipment; that is, will it withstand the continued reeling and unreeling action required of many cables, and will it withstand this action without cracking the outer jacket?

Answer: The answer to this question is an unqualified "Yes."

Question: Compared to prewar cables, will the insulation of this cable protect the conductors against mechanical blows, and will the sheath withstand abrasion due to reeling and unreeling and wear over sharp edges of coal and rock?

Answer: From my observation the properties are about equal in these respects. However, there are some added advantages in that the flame-resistant cables have greater resistance to oxidation and damage from petroleum products.

Question: Does the method of insulating a splice in a flame-resistant cable differ from previous practices?

Answer: There is no essential difference.

Question: Have there been any fires reported as a result of the ignition of these cables?

Answer: Yes. But after an investigation, it was found that actually no flame was observed by any of the workmen who were present. A composite story related by the workmen was as follows: That shortly after the installation of an approved trailing cable on a "rubber-tired buggy," the new cable fouled in the spooling mechanism of the buggy when traveling in an inby direction. When this occurred, the momentum of the vehicle violently jerked the cable pulling apart. The break was repaired and about 30 min. later, the cable failed at another point. After repair, the cable operated satisfactorily for the remainder of this shift. Two additional failures of this cable occurred during the subsequent or night shift.

On the following day, the buggy stalled near the loading machine. After about 30 min. of attempting to move the vehicle under its own power, it again started traveling slowly toward the "car-loading" point. The buggy operator noted that the trailing cable was very hot and exuding gases at a point beginning approximately 200 ft inby the fused power tap. The operator reeled up the hot cable while traveling to the loading point, stopped the machine, and removed the power tap. He then proceeded to notify the maintenance mechanic. During his absence other members of the mechanical-loading crew smelled the products evolved from the heated cable. On investigation, they found these products coming from the reel assembly of the buggy. They pulled the hot cable off the reel and cooled it with water.

The direct current at this mine is generated at 275 volts. The approved trailing cable used on the buggy was a two-conductor concentric No. 6 and the tap fuse was 200-amp rating.

Apparently four things were wrong in this particular case. First, the threading or reeling mechanism was faulty causing the cable to foul or jam, which imposed sufficient stress to pull the cable apart and obviously damage the insulation between the conductors. The latter is indicated by the three reported failures before the heating occurred. Second, the conductors were too small to handle normal current demand. Third, the tap fuse was of too high a rating. Fourth, the hot cable should not have been reeled up.

The use of No. 4 three-conductor cable was recommended; also, that fuse no larger than 80 amp be used, and, if a cable should heat, it must not be reeled up but stretched on the mine floor.

Safe Timbering of Working Places in Mobile Loading

By E. F. Miller*

The subject "Safe Timbering of Working Places in Mobile Loading" is one that requires an analytical study of the strata both above and below the coal seam as well as the characteristics of the coal seam itself. The projection or mining plan must be based on economic roof control in which the application of timber can be minimized without incurring cost increases in other phases of production. The several types of loading machines in common use today are not adaptable to the same projections or timbering systems. Consequently the type of equipment to be used must also be considered. For these reasons the writer does not believe that the problems of timbering in working places can be discussed without simultaneously considering economic production as well as the safety of men and the conservation of equipment.

Prior to the introduction of loading machines as a substitute for the pick and shovel system of loading coal, the care and maintenance of the individual working places were generally the responsibility of the coal loaders who worked in them. Up to that time very few coal companies had developed and applied fixed and definite standards for timbering the working places in the mines they operated.

It is common knowledge that mechanical loading of coal has had several false starts in the last twenty-five years and it is only in the last decade that it has developed to the stage at which we now find it. During this twenty-five year period the management of the more responsible companies have come to realize the tremendous importance of roof control in coal mining. From a safety viewpoint, the lack of roof control has been and still is responsible for more than a third of the industry's annual fatality experience. In recent years workmen's compensation and insurance rates have steadily increased while the profit margin has decreased almost to the point of elimination. From an economic operating viewpoint, the failure to properly control roof is quite often the most costly single phase of coal mining.

Since the manufacturers of loading machine equipment have so improved their machines that normal care and maintenance can reduce to a minimum the time lost because of mechanical failure the production delays from other causes have assumed a far greater importance. Roof control in the broad sense and in the light of concentrated mining plans or projections that are adapted to mobile loading should be regarded as one of the major problems in the inseparable phases of production and safety of the workman and equipment at the working face.

The subject of this paper cannot be treated properly unless what is said here is based upon the broad and fundamental problem of general roof control. The successful solution of a problem arising in any industry must have its origin in a complete analysis of the known facts or factors that pertain to it. The several illustrations

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that appear here as a part of this article are photographs that were taken in a mine operating in the Pittsburgh No. 8 seam in the Fairmont region of Northern West Virginia. In this area of the West Virginia coal fields, proper roof control has been a major factor in the successful production of coal from that seam.

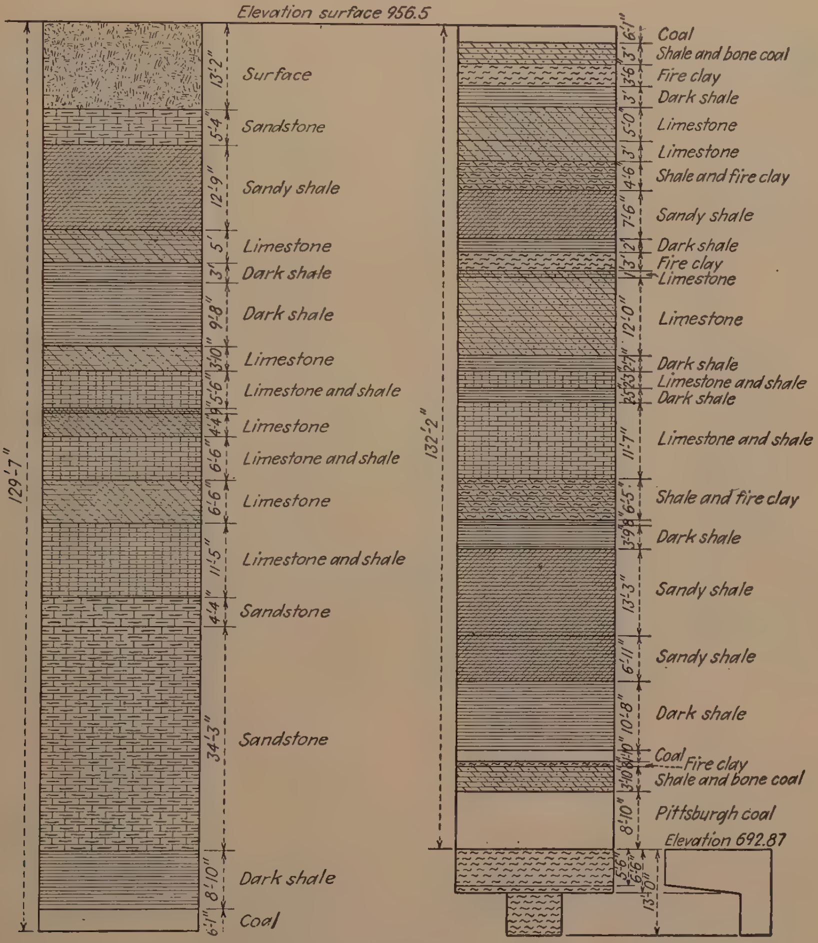

FIG I—CHART SHOWING PITTSBURGH NO. 8 SEAM OVERBURDEN.

ANALYSIS

For convenience in presentation the analytical steps are enumerated in the preliminary study of a roof control problem: 1. Columnar consist of overburden. 2. Structural strength of the coal seam to be mined. (a) Factor of resistance to the disintegrating action of ventilating air currents. 3. Beam strength of the strata composing the immediate roof. (a) Maximum width of primary entries or rooms consistent with taking full advantage of the beam strength of the immediate roof, to the end that a minimum of timbering will be required. 4. What portion of the total overburden must be supported by face timbering. (a) Over what period of time will proper roof control depend on face timbering. 5. What center interval in first mining would be most advantageous to insure support of the roof without movement of the overburden and at the same time provide the maximum percentage of total extraction.

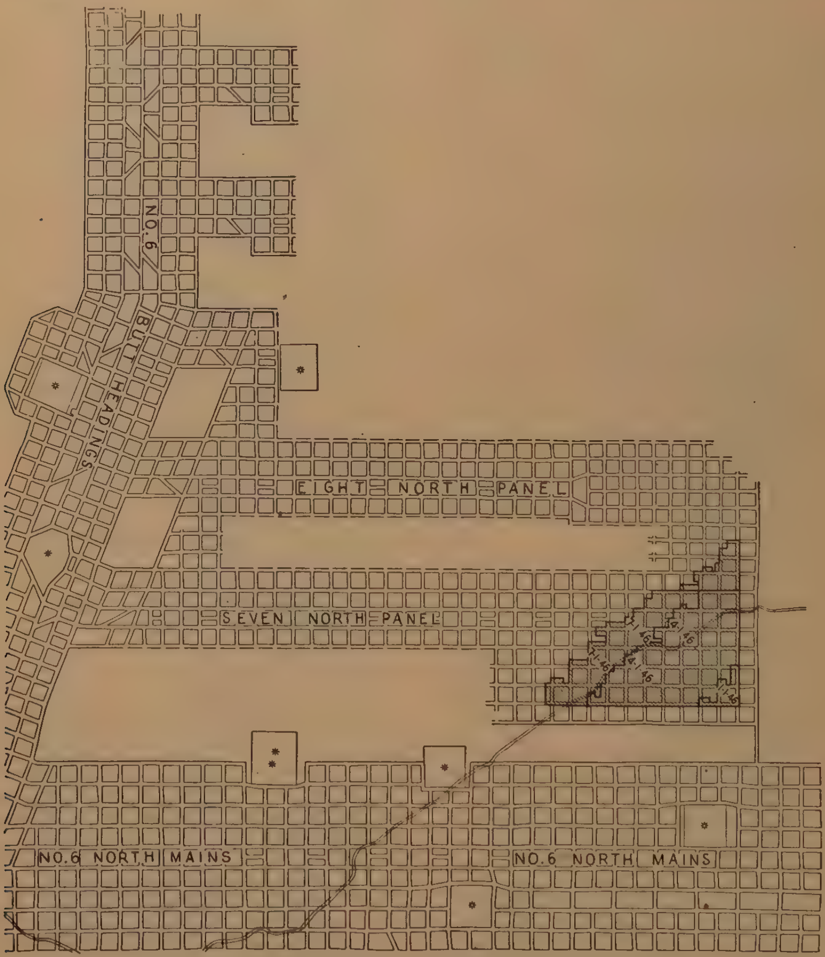

Fig 2—Active section, Pittsburgh No. 8 seam.

All the above items may be properly classified as essential to the projection for the mining of any coal seam. The relation of mining projection and development to roof control in working places is a very close one. Consequently the development of standard plans and timbering procedures for working face application requires close attention to projection plans, as well as necessitating extensive research into the physical characteristics of the coal seam, and the strata overlying and underlying the seam.

Fig 1 is a chart showing the Pittsburgh No. 8 seam overburden as it was taken from a record of shaft excavation recently made at a Northern West Virginia mining operation.

While the overburden ranges from the outcrop to approximately 600 ft, this section is somewhat characteristic of the general area and represents an average overburden.

The seam is shown as 8 ft 10 in. thick but actually averages about 8 ft 3 in. The seam contains several thin bands of bone coal, one of which is about 1 in. thick and is uniform in occurrence with relation to the top of the seam. The seam is underlaid with some 2 to 4 in. of hard shale over a 13 ft bed of low grade fire clay. Because of the expansive nature of the fire clay after it has been exposed to air or water, this 13 ft bed of fire clay has a distinct bearing on the projection of the mine and on the safe timbering of the working faces.

Immediately above the coal seam is a 3 ft 10 in. thickness of what is designated as shale and bone coal; 10 in. of soft fire clay; 1 ft 10 in. of rider or "wild" coal and 10 ft 8 in. of dark shale. The occurrence of the rider or "wild" coal is not uniform and its disappearance presents one of the most serious problems in roof control. The remainder of the overburden will not be discussed, other than to state that the strata have sufficient strength to support the overburden over fairly wide areas, but will break "clean" after the pillars and supporting timbers have been removed. Settlement and pillar "cracks" on the surface are easily discernable along a good pillar line regardless of the height of the overburden.

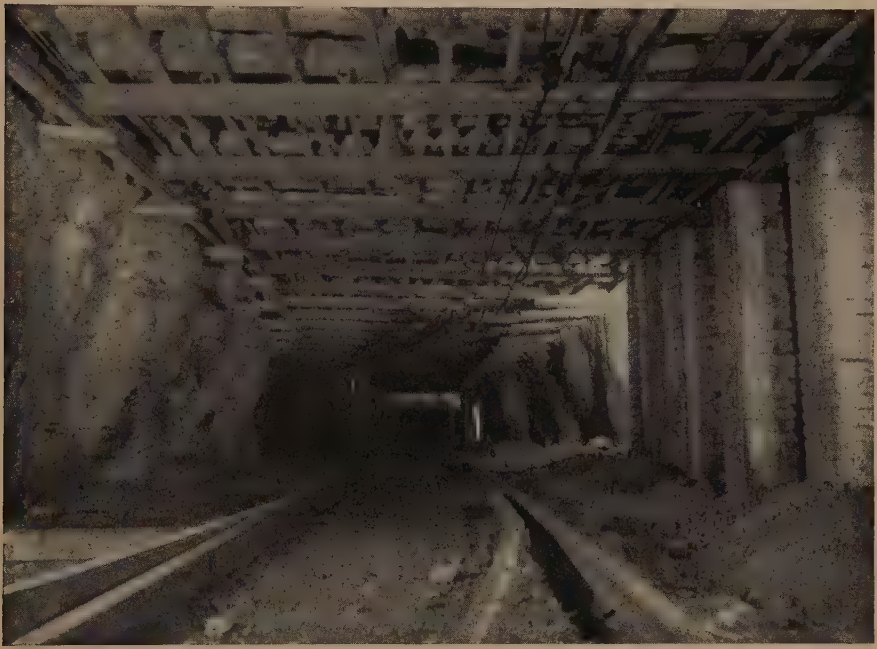

FIG 3—MAIN LINE TRACK SHOWING ROOF CONDITIONS UNSUPPORTED BY COAL.

The operation of the Pittsburgh No. 8 seam has been a major industry in Northern West Virginia for the past half century and, by trial and error, the block system retreating on a 45° pillar line has become the predominating system where maximum recovery is expected. The maximum distance between break-throughs permitted by West Virginia State Law, is now 80 ft. In other words, 80 ft is the maximum length of a side of a square block which may be left for retreating when the block system is used.

In Fig 2, which is a map of an active section in the Pittsburgh No. 8 seam, the

7 North Panel was developed as a five entry system on a center interval of 90 ft for entries as well as crosscuts. This panel was developed to a depth of approximately 3000 ft from its intersection with the main entry system. In retreat mining four additional headings were driven and the pillars extracted as indicated. The far end of 8 North Panel and 9 North Panel were developed as six-entry systems on a center interval of 80 ft which will increase the percentage of coal from first mining from 30½ to 34 pct. Only a sufficient number of places are advanced in first mining to provide for adequate ventilation and haulage. This provides for the limited amount of open area in which the roof must stand or be supported.

Since the life of a panel seldom exceeds two and one-half years it has been found that a coal roof will usually support itself in normal seam conditions in a 15 ft wide place until actual pillaring has begun. The width of the place and the percentage of coal extracted in first mining is a question for the individual operation, but experience at this particular mine indicates that approximately 34 pct of the seam is all that can be extracted in first mining when maximum recovery is expected.

Since the working place of today often becomes the haulway, manway or airway of tomorrow, attention is called to Fig 3 which illustrates a main line track where roof conditions were such that coal top would not support it.

In this particular section of the mine the "wild" coal which was discussed under strata was cut out by a soft draw slate. As the entry was developed it was timbered by means of 6-in. round oak crossbars 14 ft long on round oak posts. The crossbars were normally set on centers of 4 ft but during the development stage the bars became so heavily laden that they had to be reenforced with similar timber set between each two of those primarily installed. The reason for such roof action is that the coal top, without the aid of the wild coal, is usually insufficient to prevent air from attacking the upper fire clay, which upon expansion can seldom be stopped, regardless of the type of timbering.

Prior to the grading of the road bed and the construction of permanent main line haulage road, primary timbering had to be removed and the fallen debris loaded out to make way for the kind of timbering illustrated.

Because of the expansive nature of the fire clay bottom, piers are used only as a last resort, and at crosscuts, as shown in the figure. Piers have answered satisfactorily only when massive or placed well back in the rib which requires channeling. In the illustrated pin timbering a rail mounted Cardox drill was utilized to drill holes in the coal rib after which sections of No. 60 rail were grouted in place to serve as cantilever beam supports for the simple beam on which the steel crossbars were placed. The area between the crossbars and the top of the caved area was filled with such scrap material as locust, used treated timbers, and pipe.

Fig 4 shows a particular section of a haulage road which was developed during the war, or when steel was not available. Although the roof in this section of the mine is strong, the importance and the life of the entry decreed that no chances be taken in exposing the fire clay.

To guard against such emergencies zinc chloride-treated 7 × 9-in. hardwood was framed and set as shown.

Fig 5 as well as Fig 6 and 7 show more clearly the ordinary conditions found in the Pittsburgh No. 8 seam. In mining operations, 3 in. of coal is left on the bottom by undercutting machines and the coal is blasted in such a manner as to leave 10 to 18 in. of coal for immediate roof. This loss of top coal is not serious as the top of the seam is so high in inherent sulphur and

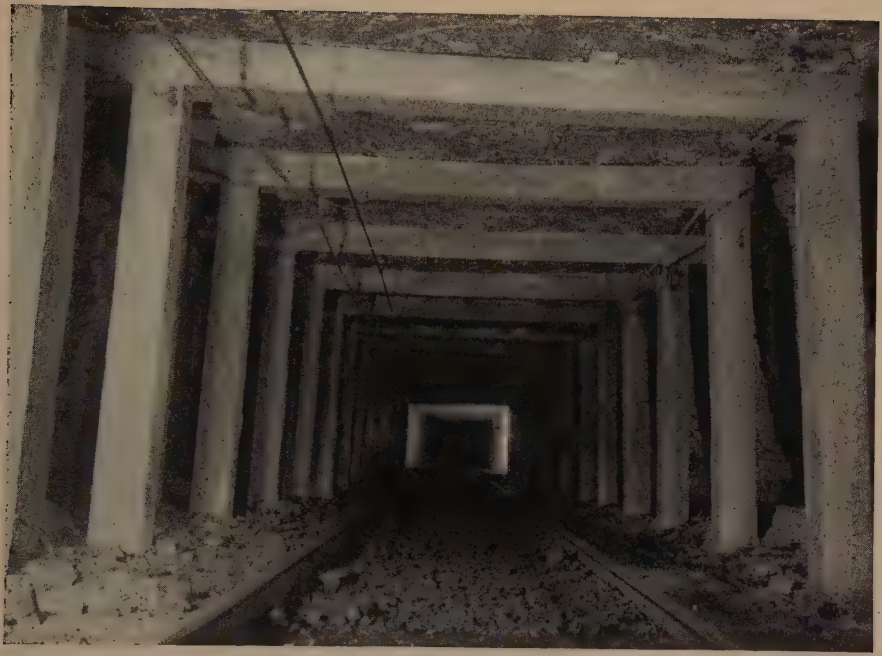

FIG 4—SECTION OF HAULAGE ROAD DEVELOPED WHEN STEEL WAS UNAVAILABLE.

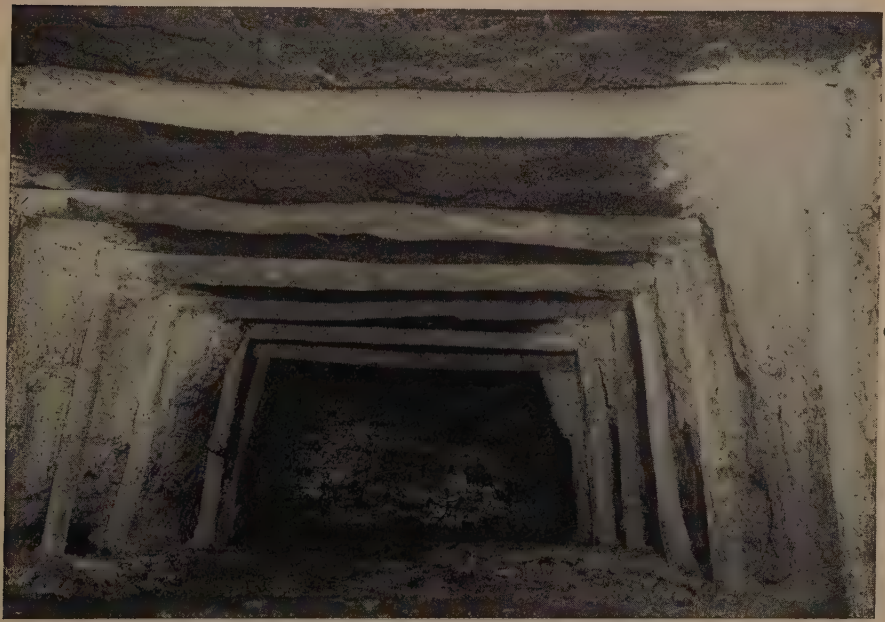

FIG 5—TIMBERED SHORT-LIFE AREAWAY.

ash that if loaded it would contaminate the remainder of the seam.

In narrow places the thickness of coal left for roof is usually strong enough in beam action to prevent the separation of roof strata in the fire clay streak between the shale and the wild coal above it. Under ordinary mining conditions, places up to 15 ft in width will usually stand without timbering for a period of three years. In sub-standard roof conditions it is often found necessary to timber short-life areaways as shown in Fig 5.

Center posting in this territory has not been found successful as the heaving action of the underlying fire clay, which is much more severe in the center of the place, will usually break the posts leaving the roof unsupported.

In 1939 and 1940 the West Virginia State Department of Mines found it necessary to make a ruling applicable to all mobile loading operations in the state. This ruling required the protection of 5 crossbars over a mobile loading unit while it is in operation. In straight line work such as some permanent haulway where few turns were necessary, the type of timbering shown in Fig 5 was found satisfactory for face timbering when it was to remain in place for the life of the entry.

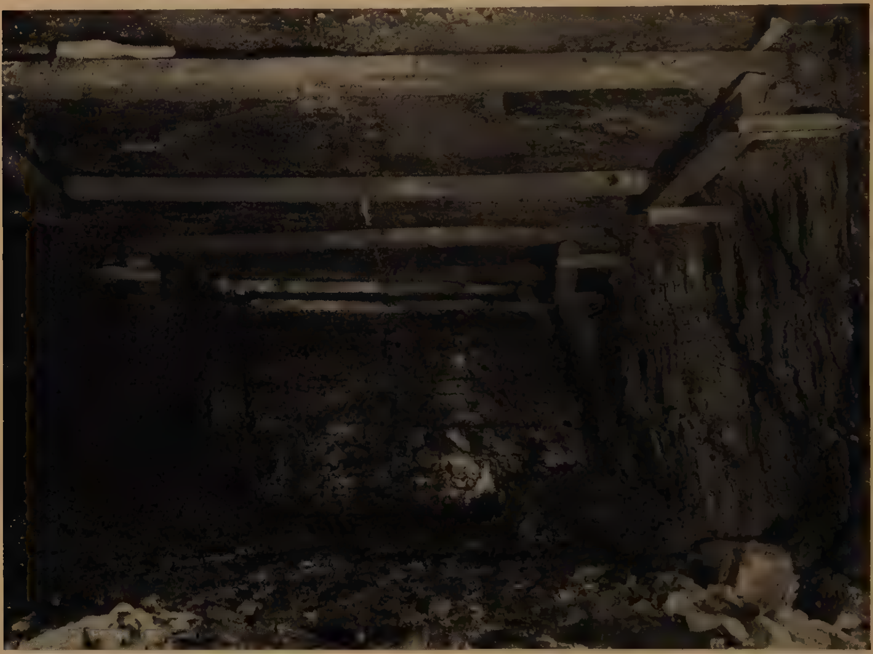

Fig 6—Face timbering for frequent turns.

For ordinary face protection in the development of narrow headings and crosscuts, with large mobile loading equipment, the posts and crossbars shown in Fig 5 are not desirable. Where frequent turns are necessary the post and crossbar become a hazard as they may be knocked out by any piece of mobile equipment. To protect the workmen and equipment from such a hazard the type of face timbering shown in Fig 6 and 7 was developed.

In this method for timbering faces the cycle of face preparation work was slightly

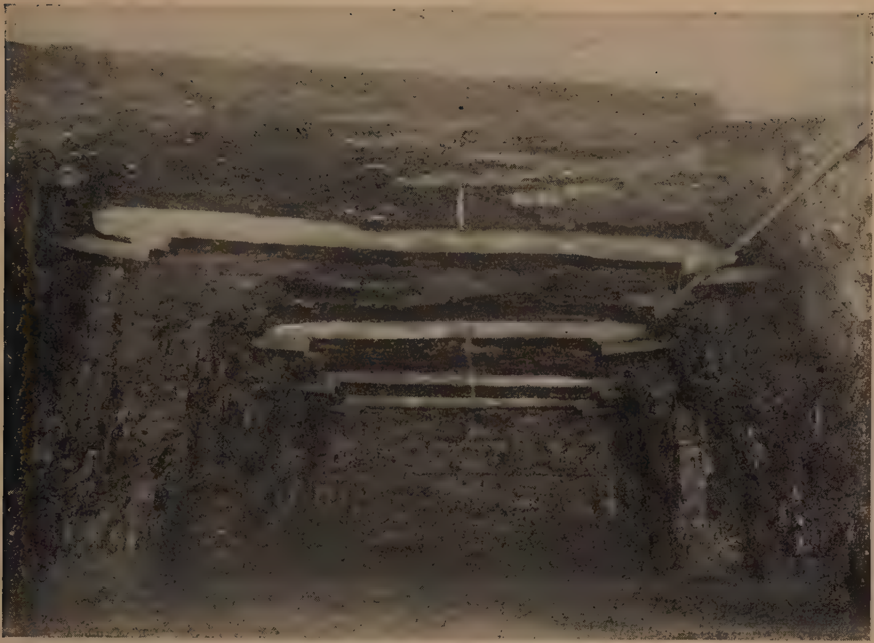

FIG 7—FURTHER FACE TIMBERING.

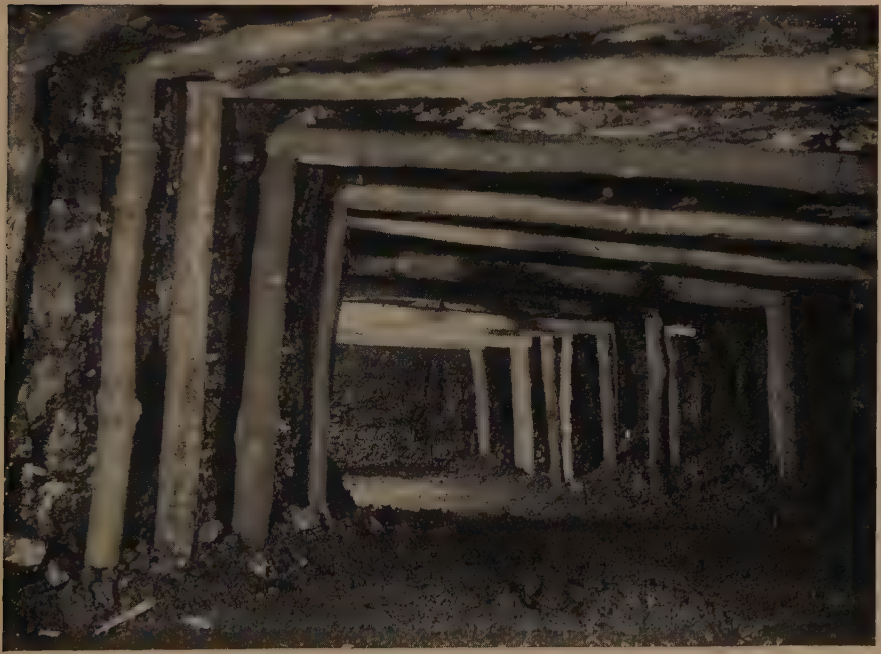

FIG 8—CROSSBAR TIMBERING IN A PILLAR LIFT.

changed so that the drilling for blasting preceded the operation of the cutting machine. The drillers were provided with a short drill and drilled four shallow holes in the ribs after each successive fall of coal had been loaded. A pin of the type shown here was then inserted in each of the four rib holes. An ordinary mine tie was then placed across each pair of pins at right angles to the face of the coal, after which the outby pair of crossbars was taken down and set as indicated adjacent to the face. Two kinds of pins will be noted in this photograph. In the experimental stage the pins were cut from 1½ in. round common steel stock. After these pins had been in use for some time it was noticed that the wedging of the crossbars was bending them to the extent that they had to be brought to the surface for straightening. Instead of buying more stock to cut into pins, discarded No. 30 steel rail stock rounded on one end to fit an ordinary coal drill hole was used. This proved to be a more suitable type of pin for temporary face timbering application.

In order to further reduce the labor and material cost of temporary face timbering in development areas, the kind of pin crossbar support shown in Fig 7 was designed and built. It consists of a 36-in. section of No. 30 rail, 18 in. of which is rounded to fit an ordinary coal drill hole with a U shaped 3 × ½ in. steel bracket welded to the flange of the rail on the opposite end. In applying this type of crossbar support only one bar is set per cut of coal which is equivalent to an interval of 8½ ft. Only one drill hole is required in each rib per cut. 4 × 6-in. × 14 ft crossbars are set over the brackets. The bar is held tightly against the roof by means of wedging between the bracket mounted on the pin, and the crossbar. In this type of pin support for crossbar face timbering care must be exercised as to the placement of the holes close enough to the roof to prevent the turning of the pin in the hole after the rectangular shaped crossbars are placed in the brackets preparatory to wedging them tightly in place. This method of face timbering has eliminated all accidents resulting from the knocking out of posts by moving equipment.

Fig 8 illustrates crossbar timbering in a pillar lift. The photographer stood in one of the excavations made in first mining and the picture shows the pillar face which has been made by successive cuts off the back end of one of the blocks which was left after first mining. On the 90 ft center projection these blocks are 75 ft square and require 3 lifts off the back end of each block to remove the pillar. On the 80 ft center projection the blocks are removed by driving two lifts through the pillar. In driving the lifts care is required to leave intact approximately 2 ft of coal between the working face and the caved area. This fender provides additional protection to the workmen and also prevents the contamination of the coal by the debris. 6-in. round crossbars on 6-in. round posts are set on 4 ft intervals as the face advances, for the protection of men and equipment.

To insure the prompt caving of the place after the pillar lifts have been driven through to the gob area, crossbars are pulled by means of a shuttle car and steel cable. Normally, 50 to 60 pct of the round crossbars are recovered in this operation and are used again. Generally, all the posts are lost when the roof fall is made.

In this general plan of mining, systematic caving of the mined area is an absolute essential to successful roof control in subsequent and adjacent mining. When the coal has been extracted from a pillar lift and the temporary timbering drawn, the overburden is fractured along the line of intersection between the unmined coal and the mined area. Thus the pressure of the overburden on the unmined coal seam is relieved in much the same manner as the pressure on the supporting member of a cantilever beam is relieved when the beam fails at the point of support. In the event of failure to fracture the overburden as the pillars are extracted, the overburden will subside in a bending action developing what is called "rides" or "squeezes." If this occurs, many working face roof control problems are induced. Quite often the roof control problem may become so expensive and hazardous as to make it necessary to stop the mining of coal in the area in which the "squeeze" occurs. In the general system of mining outlined here, an average of 90 pct extraction of the seam may be expected.

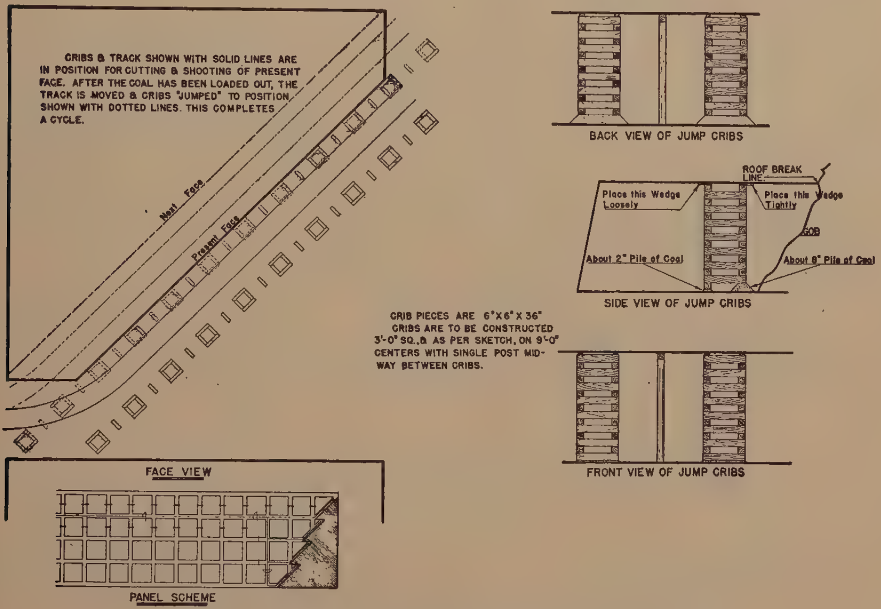
CRIBS & TRACK SHOWN WITH SOLID LINES ARE IN POSITION FOR CUTTING & SHOOTING OF PRESENT FACE. AFTER THE COAL HAS BEEN LOADED OUT, THE TRACK IS MOVED & CRIBS "JUMPED" TO POSITION SHOWN WITH DOTTED LINES. THIS COMPLETES A CYCLE.
Next Face
Present Face
BACK VIEW OF JUMP CRIBS
ROOF BREAK LINE
Place this Wedge Loosely
Place this Wedge Tightly
GOB
About 2" Pile of Coal
About 6" Pile of Coal
SIDE VIEW OF JUMP CRIBS
CRIB PIECES ARE 6"X6"X36" CRIBS ARE TO BE CONSTRUCTED 3'-0" SQ., & AS PER SKETCH, ON 9'-0" CENTERS WITH SINGLE POST MIDWAY BETWEEN CRIBS.
FRONT VIEW OF JUMP CRIBS
FACE VIEW
PANEL SCHEME

Fig 9—System of pillaring with loading machines.

Numerous attempts have been made to improve upon the method and safety of recovering pillars in the Pittsburgh No. 8 seam of coal

Naturally greater safety could be attained at the expense of extraction but it is questionable if our country can continue to waste its natural resources to the extent that it has been doing for the past century.

Fig 9 represents some sketches of a system of pillaring with loading machines used in one mine in the Pittsburgh No. 8 seam in 1923.

Working face roof control was primarily accomplished by temporary crib lines which were taken down and reset as the pillar working face was retreated. Whether or not the terminology was correct, these temporary cribs were called "jump" cribs. The descriptive term "jump" probably had its origin in the movement of the crib blocks when the key block was removed to take out the crib. The blocks would spring some 8 to 10 ft toward the retreating working face upon removal of the key block. Of course, the reason for this was the construction of the crib and the subsiding action of the overburden.

At the stage of retreat mining indicated in the detail of the working face in this figure, two cribs adjacent to the gob area on the new line were set before any of the cribs in the preceding location were taken out. Then the No. 1 crib on the old line was taken down and the crib blocks reused to build the No. 3 crib on the new line. This procedure was repeated until all the cribs on the old line had been taken down and reset in the new location as illustrated. If the weight of the overburden had not been such as to compact a pile of fine coal, used as a foundation for the superimposed crib blocks on the gob side of the crib, the crib could be "jumped" by removing part of this pile of coal with a shovel, while standing in a position near the post that was set between each two tiers of crib blocks. In the event the pile of slack at the base of the rear side of the crib had been too tightly compressed to be easily removable, the crib was "jumped" by dragging the point of a pick through the 2-in. bed of slack that was used as a foundation for the bottom crib block on the front side of the crib. The downward pressure of normal subsidence would then dislodge the remaining blocks of the crib in much the same manner as the pressure of a thumb and forefinger removes the edible portion of a blue grape from its skin. The posts set between the tiers of crib blocks were left in place and were subsequently broken by roof subsidence.

This method of face timbering, of course, required very close supervision but as a reward, provided high unit production and maximum extraction.

Transportation Hazards—Causes and Prevention

By Andrew Hyslop, Jr.*

(New York Meeting, February 1948)

In our never ending search for new and better ways of underground mining, we find that transportation has had its share of new ideas in the past few years. The old and still effective method of track haulage has had some competition from rubber-tired cars and belt conveyors, each of which has its own safety hazards which demand close scrutiny in any safety study. The use of shuttle cars, in particular, has introduced clearance hazards caused by the absence of the more or less positive guides formed by the rails of a track system; likewise the requirement of greater clearance tends to increase roof hazards which we all know to be great enough with the best of methods. However, the subject of transportation hazards is too broad to be covered entirely in one paper, so I will confine myself to track methods which carry the preponderance of underground tonnage.

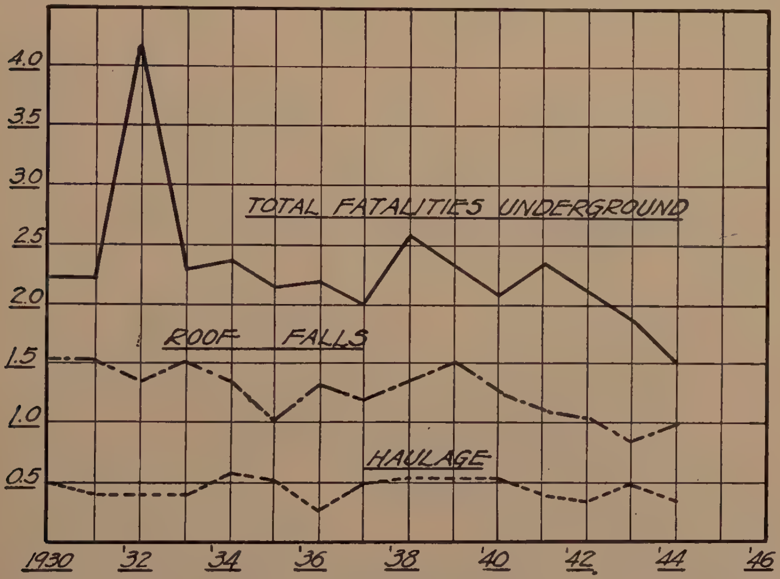

Fig 1—Illustrating for Illinois, a slight downward trend in fatality rates for total fatalities and roof-fall fatalities.

I am quite sure that to this group it is an evident fact that haulage accidents are of major importance. However, to stress the point, it may be well to look at a few figures. Safety statistics can be quite boring and often are subject to question as to mathematical accuracy because of different

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methods of compilation and computation but a few are presented to give an approximation of the values involved in our subject.

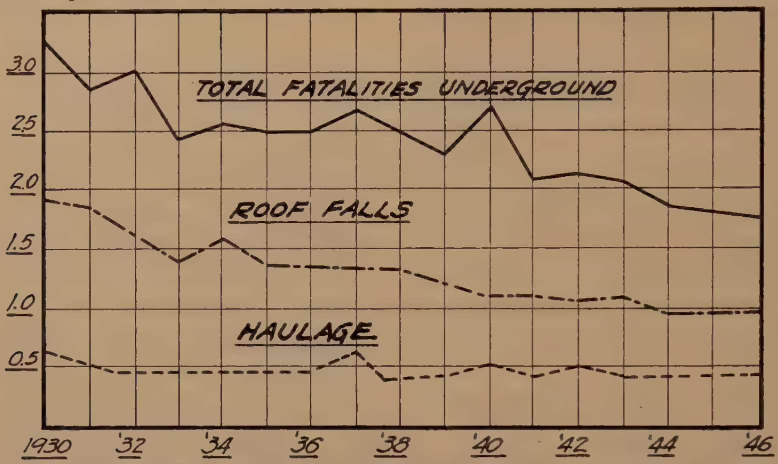

Fig 2—Illustrating the fatality rates nationally for all bituminous coal mines.

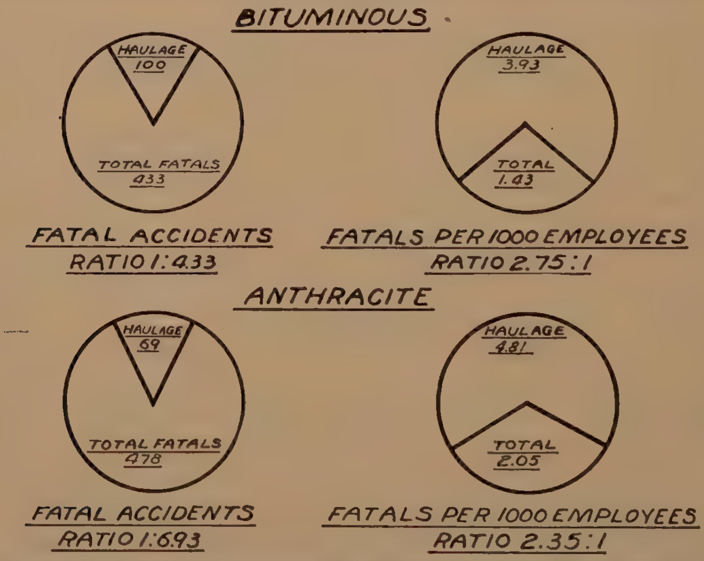

Fig 3—Indicating that haulage fatalities accounted for 23 pct and 14.5 pct respectively of total fatalities in the bituminous and anthracite mines of Pennsylvania during 1944–1946, while on an exposure basis haulage employees have a much greater liability to fatal accidents than other employees.

Fig 1 shows for the state of Illinois a slight downward trend in fatality rates for total fatalities and roof-fall fatalities, while the fatality rate for haulage remains essentially constant for the past 14 years.

Fig 2 illustrates the fatality rates nationally for all bituminous coal mines. The trend of roof fall fatalities is definitely downward while the haulage fatality rate again remains constant.

Fig 3 indicates that haulage fatalities accounted for 23 and 14.5 pct respectively of the total fatalities in the bituminous and anthracite mines of Pennsylvania during

1944–1946. However, when the fatality rates are compared on an exposure basis, the frequency rates for haulage employees are disproportionately higher than for other employees.

Fig 4 gives the comparative ratios for fatal and non-fatal accidents against the frequency ratios for roof falls and haulage in the state of Illinois.

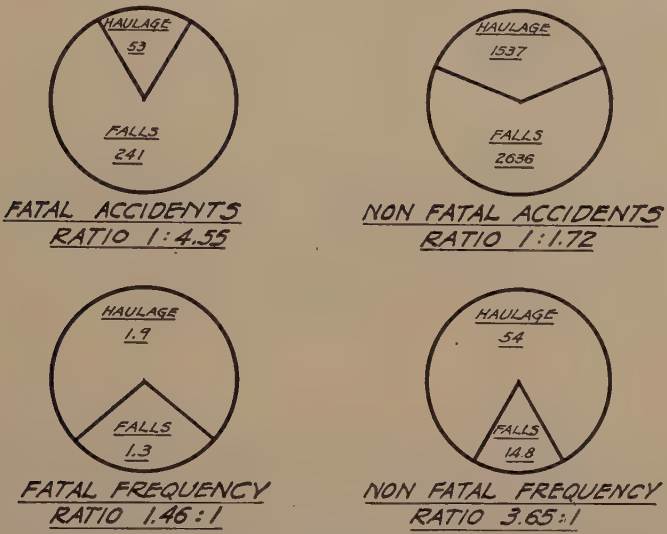

FIG 4—COMPARATIVE RATIOS FOR FATAL AND NON-FATAL ACCIDENTS AGAINST FREQUENCY RATIOS FOR ROOF-FALLS AND HAULAGE IN ILLINOIS.

Fig 5 is a comparison similar to Fig 4 for the state of West Virginia. The haulage frequency in non-fatal accidents again predominates.

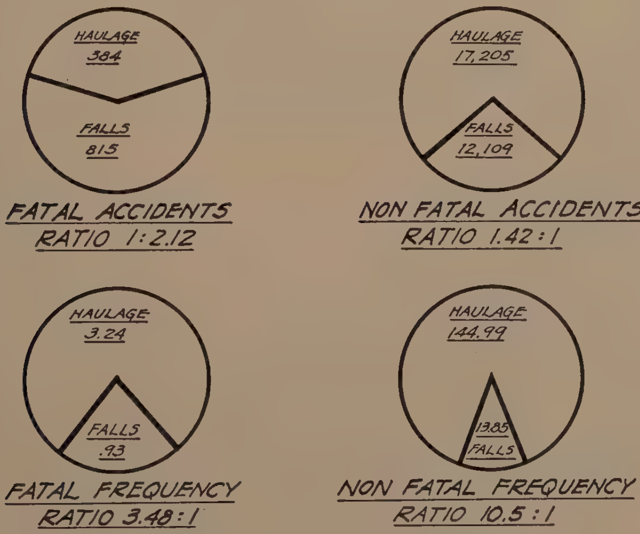

FIG 5—COMPARISON SIMILAR TO FIG 4 FOR WEST VIRGINIA.

Knowing these facts, the mining indus-

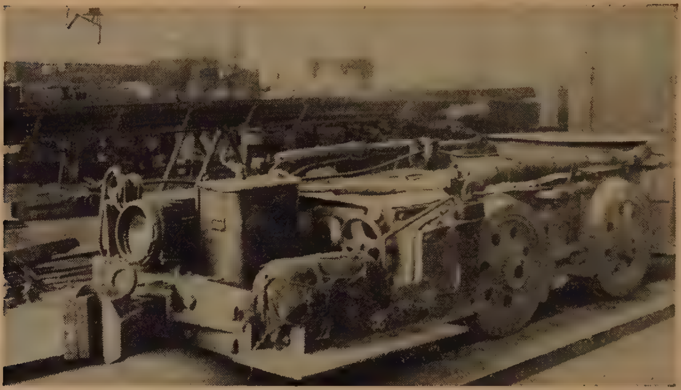

FIG 6—A FAMILIAR SIGHT IN MOST MINES. THE CONVENTIONAL MINE LOCOMOTIVE ESSENTIALLY THE SAME BASIC DESIGN AS MOST LOCOMOTIVES STILL BEING MANUFACTURED. THIS IS A DELUXE MODEL WITH AN UPHOLSTERED SEAT.

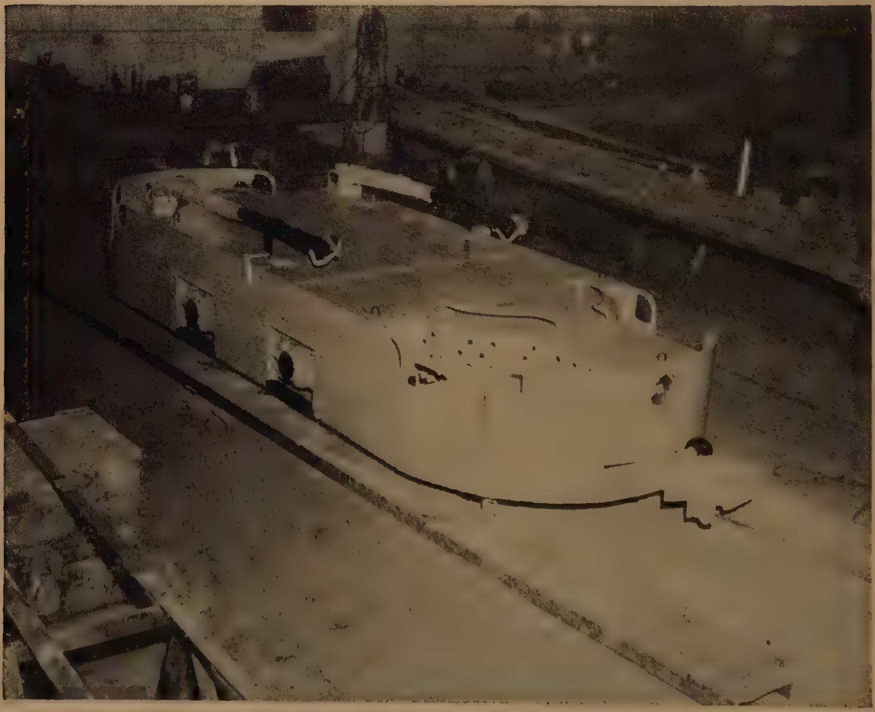

FIG 7—THE SAME TYPE LOCOMOTIVE AFTER BEING REBUILT IN OUR SHOP. THE SQUARE END BUMPERS ARE REPLACED WITH SEMICIRCULAR PLATES ON BOTH ENDS OF THE LOCOMOTIVE. NOTE THE SEATS PROVIDED FOR THE TRIP-RIDER ON EITHER SIDE OF THE LOCOMOTIVE.

try is compelled to give very serious consideration to reduction of haulage accidents.

In safety work in the mines, it is very easy to get a warped perspective on the relative values of contributing causes of mine accidents and injuries. For example, much national publicity is given to major mine disasters which at times create a furore in our Federal government from the executive branch down and start a flood of adverse criticism of the mining industry, resulting in endless revision of safety regulations and practices at a cost to the mining industry of millions of dollars annually. Yet haulage accidents result in approximately twice as many fatalities as major disasters, plus a great number of non-fatal injuries, and are given only comparatively meager, haphazard attention. To accomplish results in our safety efforts, we must obtain first a clear view of the values involved. When a fair analysis is made, haulage accidents obviously demand serious attention in our safety program.

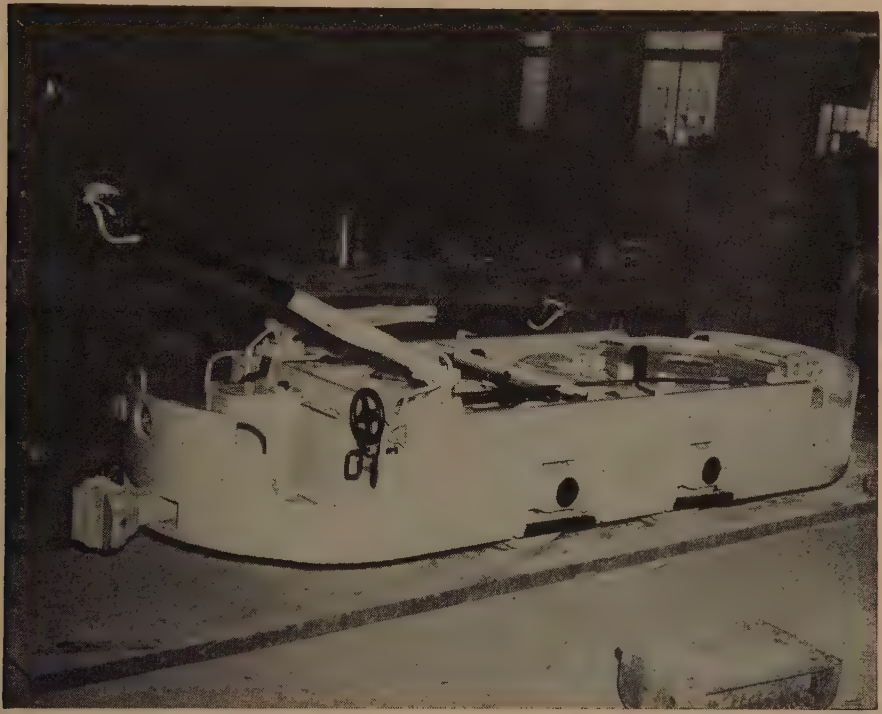

Fig 8—The cab end, covers removed from top of locomotive. Note cut-away sections on either side for easy access to the cab. Convenient grab irons are provided and cab is large enough to seat two men comfortably.

It is my opinion that in haulage accidents as in all other types of accidents, proper safety training of all employees will do more than anything else to reduce accidents. A preponderant majority of injuries is the result of repeated risks taken by the workman, and the inexorable law of averages takes its toll. A survey of general industrial accidents made by a large insurance company showed that for every major injury, that is, fatality, fracture, dismemberment, or other serious injury, there are 29 minor injuries and 300 accidents without injury, all from the same cause. These 300 accidents without injury, give us a most fertile field for constructive work in accident prevention. To anyone who has observed underground haulage with even casual attention, it must be evident that in the average mine there are many acts of the haulage personnel which make accidents and injuries quite probable. To mention a few: Unnecessary pushing of cars, backpoling, coupling and uncoupling moving cars, coupling on tight side, riding between cars, getting on or off moving trips, flying switches, improper blocking, obstructed clearance, lack of trip lights or markers, and poor condition of track, cars and locomotives.

The list of haulage accident possibilities is quite long even for the best of mines and in spite of voluminous rule books. It is my belief that if any marked improvement is to be made in our haulage record, methods must be developed to stop unnecessary risks which exact a price for accidents and injuries in a ratio similar to the 1:29:300 mentioned above.

It is not within the scope of this paper to present or outline safety programs, but to reduce haulage accidents in any worthwhile degree, a first essential is that there be a working safety program that engenders safety-conscious work habits in each workman. Such a program must of necessity start with a whole-hearted policy on the part of top management to keep safety ever foremost in the minds of all employees. The foreman should be a key figure in this program. He is the man in close everyday contact with the worker. He should represent management to the worker, and, if he is competent in his supervisory work, should be able to get effective safety cooperation from his men. A competent safety engineer is very necessary to school the foreman in his safety work. I say

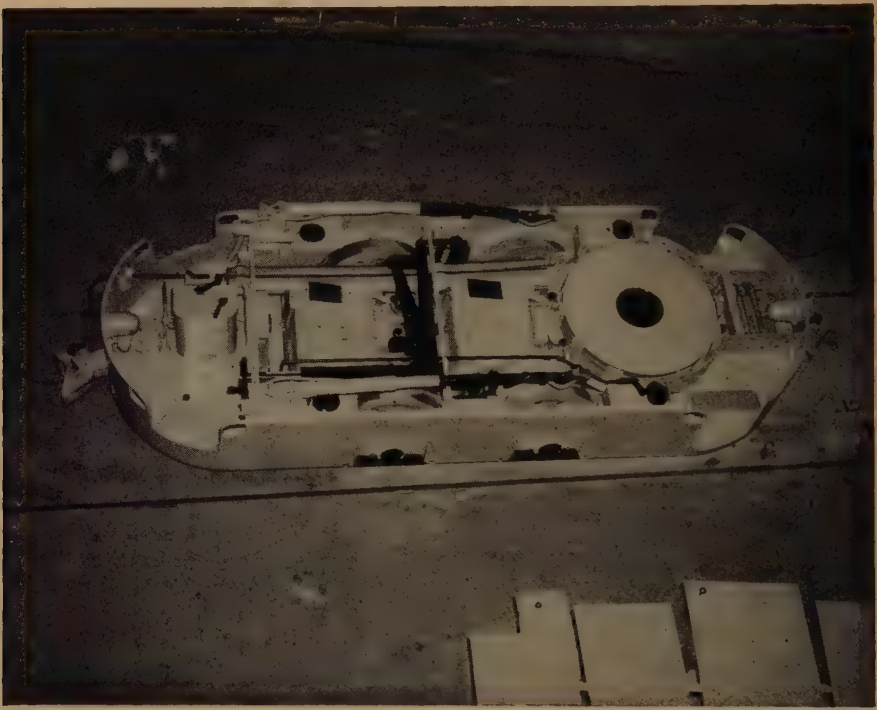

Fig 9—Bird's eye view with covers removed showing general arrangement of units Cable reel is below the top of sideframe. Note large cab compartment.

engineer rather than inspector because our mines are now burdened with overlapping, time-consuming inspections which do little more than point out problems and difficulties of which the mine management had already been aware. Our mines need constructive engineering assistance to help the foreman overcome these recognized problems and hazards. The most important function of the safety engineer is to help the foreman train his men in safe working habits. A quiet word of caution from the foreman to a trip rider or snapper getting on the locomotive improperly or coupling moving cars usually will produce more effective and lasting results. Safety meetings conducted by the foreman in which accidents are discussed frankly and honestly are of great value in developing safe work habits. To do this, the foreman must be well schooled in the art of conducting conference-type meetings, and the burden of this schooling rightfully falls on the safety engineer.

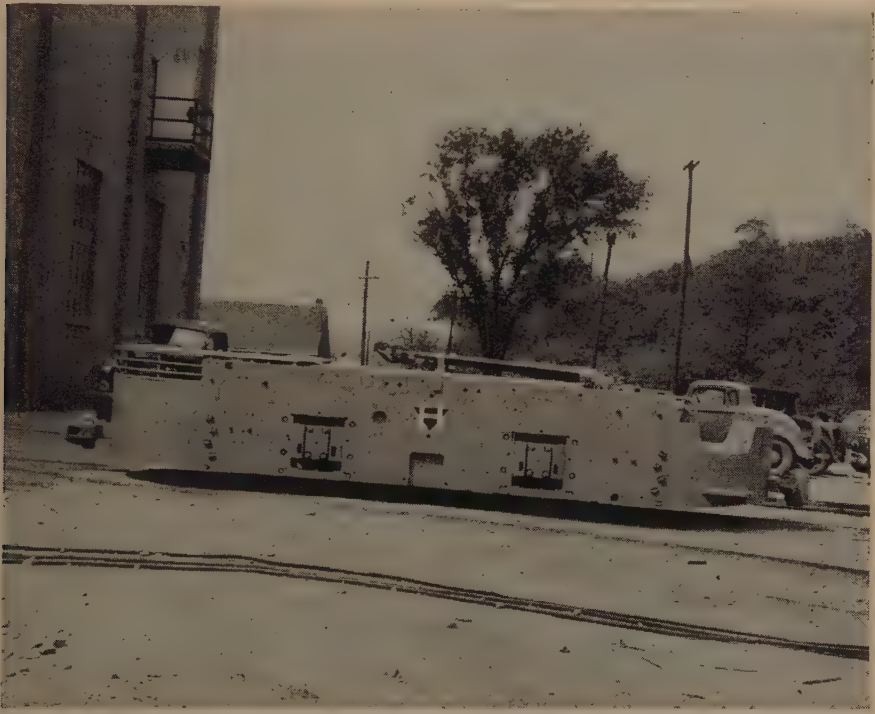
FIG 10—A 13-TON HAULAGE MOTOR WITH THE SAME TYPE CURVED ENDS. NO TRIP-RIDER'S SEAT ON FRONT END AS HE IS SUPPOSED TO RIDE IN CAB WHICH IS OF AMPLE SIZE. FRONT CURVED PLATE PROVIDES WELL-VENTILATED COMPARTMENT FOR RESISTOR AND ELECTRICAL PARTS. SPACE IS PROVIDED FOR RE-RAILERS IN FRONT END UNDER HINGED LID.

Studies of accident records indicate that if all known safety features were applied to our mining machinery and equipment, there would be only a 15 to 20 pct decrease in our accidents. In other words, the bulk of our accidents are caused by human failures or by a lack of safety mindedness on the part of our workmen. It follows then that any effective safety program must strike at this basic difficulty. However, it is of paramount importance that the safety program be backed by an honest effort to make the working conditions and equipment as safe as possible for "what you are thunders so loudly I cannot hear what you say."

Transportation includes the hoisting of coal and material in shafts, conveyor inclines, and rope operated inclines, which, although they possess inherent safety hazards, account for a minor portion of our haulage accidents. Studies show that about 75 pct of haulage accidents involve the victims' being struck, run over, or squeezed, and these accidents occur for the most part at the face regions where the bulk of our handling of cars and equipment is done. This fact emphasizes the importance of the foreman's part in reducing haulage accidents because of his immediate supervision of this work.

A serious hazard in the average haulage system is the condition of the track. The more or less temporary nature of a mine track system encourages slipshod construction methods over the entire system, with the consequence that most mines have track conditions that are neither safe nor economical. It is my opinion that a 40-lb rail is the minimum size which should be used in the modern, mechanical mine. All secondary haulage roads with any degree of permanence should be laid with at least 60-lb rail. The road bed should be prepared with care for proper drainage, elevation and ballast. The mining system should be such that proper radii are possible on the curves. Parallel switch throws should be used on all switches. Poor track is not only a dangerous safety hazard but is also very expensive. Studies in track cost prove that good track is the most economical. I am quite sure this is evident to most practical mining men and our poor mine track is the

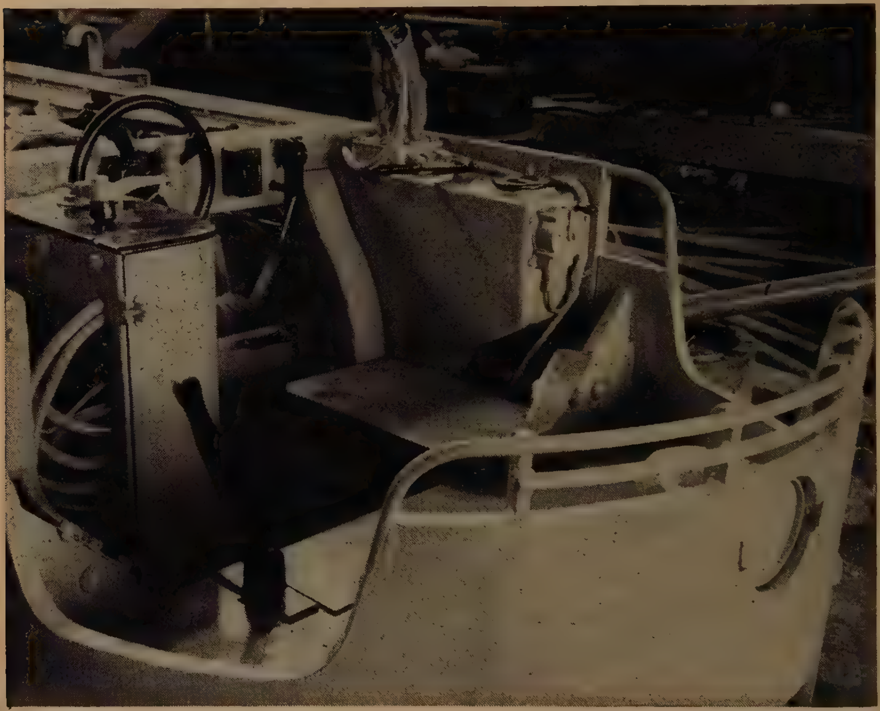

FIG 11—CLOSE-UP OF CAB SHOWING LARGE SIZE, EASY ACCESS, AND ADEQUATE GRAB IRONS. A LEVER IS PROVIDED IN THE CAB FOR UNCOUPLING EITHER END OF THE LOCOMOTIVE FROM TRIP.

result of nothing short of slovenliness and downright laziness. Fortunately, larger cars and heavier mining equipment are forcing better track, but this phase of our haulage system should be given much more consideration.

It is not too unusual to see mines in which frequent roof falls along the haulage road stop traffic and create a serious safety hazard. In timbering high speed haulage roads, it is necessary that the roof support be protected properly from dislodgement by derailments. This can be accomplished by "hitch drilling" and supporting beams on pins; or by recessing the supporting posts. I feel that the use of concrete by the "gunite" process never has been utilized to the full extent of its possibilities in sealing and supporting the roof and ribs of haulage ways. My personal experience with guniting indicates that it offers definite benefits, both in quality of the job and in economy. Usually, with poor roof support, we also find much debris along the track and piled in refuge holes and other openings. Since there often is much combustible material such as coal and timber in this debris, a dangerous fire hazard is created. The history of major mine disasters contains far too many incidents in which this debris has been the cause of, or at least a contributing cause to, the disaster. It goes without saying that such material also is a serious stumbling and clearance hazard.

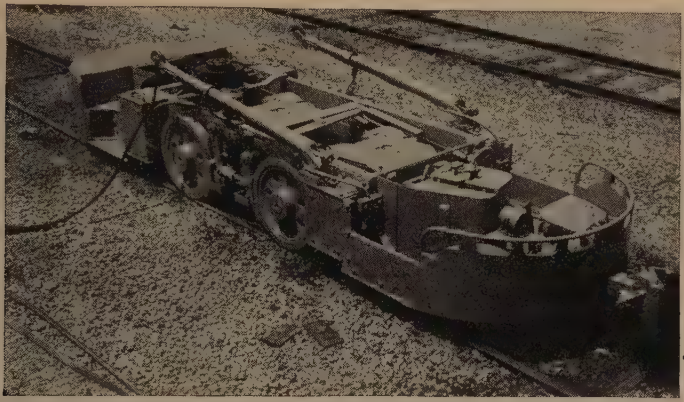

FIG 12—VERTICAL VIEW OF AN INSIDE-FRAME GATHERING LOCOMOTIVE SHOWING AGAIN THE LARGE CAB AND TRIP-RIDER'S SEAT PROVIDED ON EITHER SIDE OF FRONT END.

Good lighting along the main haulage road is another necessity. Most mine lighting is haphazard and much of it is so installed that the hazards resulting from the poor installation are greater than the benefits derived from the light. For main haulage, it is my belief that the most satisfactory arrangement is a lighting circuit entirely separate from the trolley line. It preferably should be a 110-v ac circuit supplied from lighting transformers. Such an arrangement provides much more efficient light in that the voltage can be maintained, it can be protected adequately against faults, the bulb cost is lower, and the lights remain when the dc power is interrupted.

On single track lines, it is imperative that a good system of block signals be installed. The difficulties with most signal systems in mines are not in the systems themselves but rather in the failure to enforce the rules of the system. The practice of allowing two locomotives to travel in one section at the same time is extremely dangerous and many serious accidents result from this safety rule violation. Safe practice permits only one locomotive in each block section at a time. Any other arrangement is certain to lead eventually to an accident, and quite probably a fatal one, for some haulage employee.

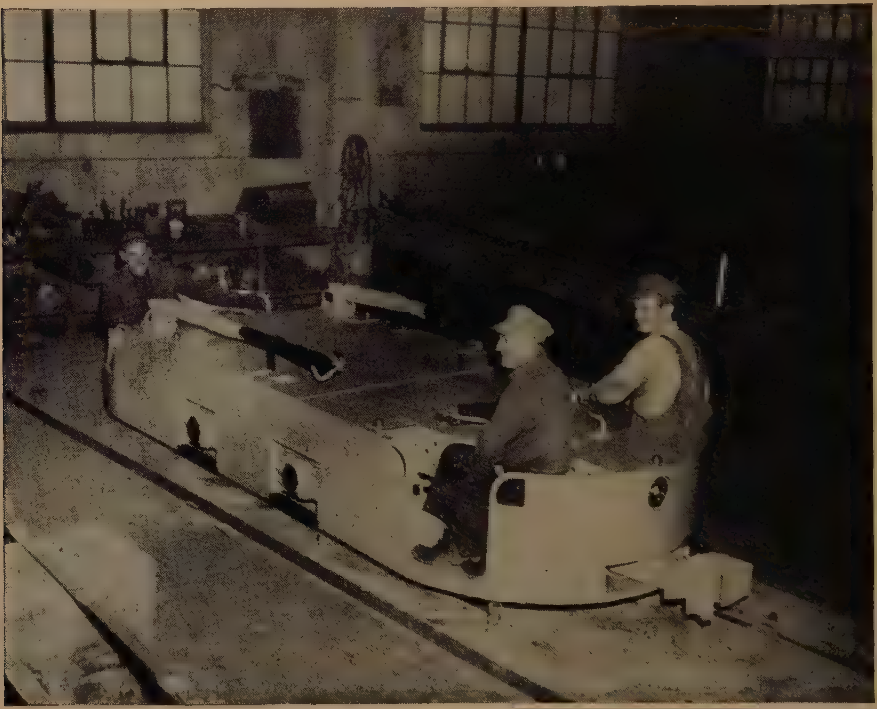

FIG 13—SHOWING MEN IN LOCATIONS PROVIDED. ALL SEATS WELL PROTECTED AND EASILY ACCESSIBLE. A NUMBER OF THESE LOCOMOTIVES HAVE BEEN IN SERVICE FOR PERIODS UP TO 3 YEARS WITH AN EXCELLENT SAFETY RECORD.

We have seen the trolley phone come into extensive use recently. This instrument can be very useful for direction of the haulage system. In case of an emergency such as a mine fire, it can be a most useful tool for safety. Instructions given to any locomotive or communication between locomotives are heard by all units equipped with the phones, thus there is a much better informed system with a consequent reduction in the possibility of collisions. However, the trolley phone only supplements the block signals. If only one system is to be used, the block-signal is preferable.

I know of no other phase of mining that is operated with as poor equipment in general as the average haulage system. Take for example, mine cars which are an important part of the haulage system. A high percentage of the mine cars in service today are of crude design. In any large group of coal miners, we always find quite a display of amputated fingers and maimed hands which bear gruesome testimony of the link and pin coupling. Fortunately, we find this crude car equipment making a slow and belated retreat, to be replaced by cars of improved design, including automatic and semi-automatic couplings. Increased car capacity also offers a considerable safety advantage. Large cars were looked upon with serious misgivings, safety-wise, when they were first introduced. Predictions were that these heavy loads could not be handled safely, thus introducing accident hazards at the face. However, it has been proven rather definitely that the large car is in fact much safer because of the considerable reduction in the number of cars to be handled for a given tonnage, and the heavy loads of necessity are handled with more care.

One of the worst examples of poor safety design is found in the average mine locomotive. The design of most present day locomotives differs little in safety features from that of thirty or forty years ago. In conventional locomotive construction, the operator and trip-rider are given very little consideration. A board bolted to the bumper usually forms an uncomfortable and precarious seat for the motorman. An accidental blow on the head or even a severe jar caused from derailment may throw him onto the track in front of the moving trip. This is a fairly common occurrence and is considered just another accident. The trip-rider, or snapper, fares little better in safety protection. If he rides on the locomotive, about the best he can hope for is an abbreviated step-plate and he is fortunate if a grab-iron is provided. Thus with most of his body outside the locomotive frame and with an uncertain hand hold by which to balance himself, he, too, often is subject to injury from squeezing or falling off the locomotive. The Hanna Coal Co. has found it rather simple to rebuild the conventional locomotive to correct these faults and make a much safer machine. The square end bumper is removed and a semicircular plate of rolled steel is attached. This plate usually extends to near the height of the locomotive frame. It thus provides protection for the operator with ample space for himself and his trip-rider if the latter rides on the locomotive. The hazard of falling out is eliminated completely and much better protection from collision hazards is provided. On the front end of the locomotive, the additional space provided by this construction makes a well-ventilated space for the resistor and other electrical equipment. We are able also to provide space for re-railers, thus keeping all equipment off the top of the machine where it is often an obstruction to vision or a clearance hazard.

With this construction on gathering locomotives, a seat is provided on either side of the locomotive for the trip-rider so that he may ride safely and comfortably. In case of derailment, the conventional square-end locomotive has a definite tendency to turn at right angles to the track if it hits a rib post or other obstruction alongside the track. This is very dangerous as the full inertia of the moving trip will be expended suddenly on the locomotive causing extensive damage to the machine and the possibility of serious injury to anyone on the locomotive. With the rounded ends, the locomotive will have a tendency to be guided parallel to the track and may bring the trip to a stop with no damage to either equipment or personnel. We have been using this construction on rebuilt locomotives for about three years, during which time we have recorded many instances where this arrangement has prevented serious or possibly fatal injuries.

Many other hazards in mine haulage have been mentioned in most of our safety-rule books and are well known to most of us. I have pointed out only a few of the most important haulage hazards that I feel should have earnest attention in our effort to make mining safer.

To allow the continuation of our bad haulage practices with their heavy toll of injuries, without sincere and practical efforts to improve the record, would be serious negligence on the part of mine management. Most of the conditions listed above are not only safety hazards but constitute poor practice from an efficiency standpoint. A well maintained haulage system with good equipment is usually an efficient one and it can be said truly that safety pays in transportation systems.

Preliminary American Tests of a Cyclone Coal Washer Developed in the Netherlands

By M. R. Geer* and H. F. Yancey,* Members AIME

(Philadelphia Meeting, October 1946)

Although the use of dense suspensions for coal cleaning was pioneered in the United States with the Chance sand flotation process, and during the past year a pilot plant using a magnetite suspension has started operation in this country, the more recent developments in this field of coal cleaning have occurred principally in the Netherlands. During the thirties the Dutch developed the Barvoys process, in which a medium of barite and clay is employed, the Tromp process, which utilizes a magnetite suspension, and the Loess process, in which is used the air-borne earth or soil known as loess.[1-3] All these processes are finding increasing use in European coal-cleaning practice. Now a fourth process, the cyclone cleaner, is under development by the Netherlands State Mines.

A cyclone thickener for reclaiming and thickening the dense suspension is a novel feature of the Loess process. Through accident, the discovery was made that the cyclone could be utilized as a cleaning device for fine coal. When the cyclone thickener at the Maurits mine plugged, the overflow was found filled with clean coal free of refuse, and from this observation developed the concept of the cyclone as a coal cleaner.

In June 1945 the cyclone cleaner was observed by one of the authors,[4] and in September 1945 it was described by Driessen[5] in a paper presented in London before the Institute of Fuel. As the cyclone seemed to offer unusual promise as a means of extending heavy-medium cleaning to the finer sizes of coal that cannot readily be treated in existing heavy-medium processes, the Bureau of Mines, following Driessen's, presentation, launched an investigation of the cyclone as a coal cleaner. Added impetus was given to this study by the present increased interest in heavy-medium cleaning in this country.

Object and Scope of Investigation

Driessen's report, although comprehensive in many respects, did not include a detailed description of the cyclone cleaner, and contained almost no information on the influence of the various operating variables of the cyclone. It was necessary, therefore, to investigate the influence of the cyclone adjustments on its performance before studying the application of the cyclone to various coal-washing problems. The present report is confined entirely to these preliminary tests, designed to demonstrate the operating characteristics of the cyclone.

The investigation has been in progress

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[1] References are at the end of the paper.

for only a few months, consequently the preliminary tests have not explored completely the entire range of operating adjustments. However, it was thought desirable to publish the results obtained thus far because of the present interest in heavy-medium cleaning.

EQUIPMENT AND TESTING PROCEDURE

Fig 1 shows the construction of the cyclone. It consists of a conical section 3⅞ in. high, and 3 in. in maximum diameter, surmounted by a cylindrical section 11/16 in. high, of the same diameter. The feed, consisting of coal and suspension, enters the cylindrical section tangentially through a ⅜-in. pipe. A removable constriction tube in the feed line provides means for regulating the feed rate. Similarly, interchangeable constriction nuts are screwed onto the bottom of the cyclone to regulate the refuse discharge. The washed-coal discharge is controlled by inserting orifice plates of suitable size.

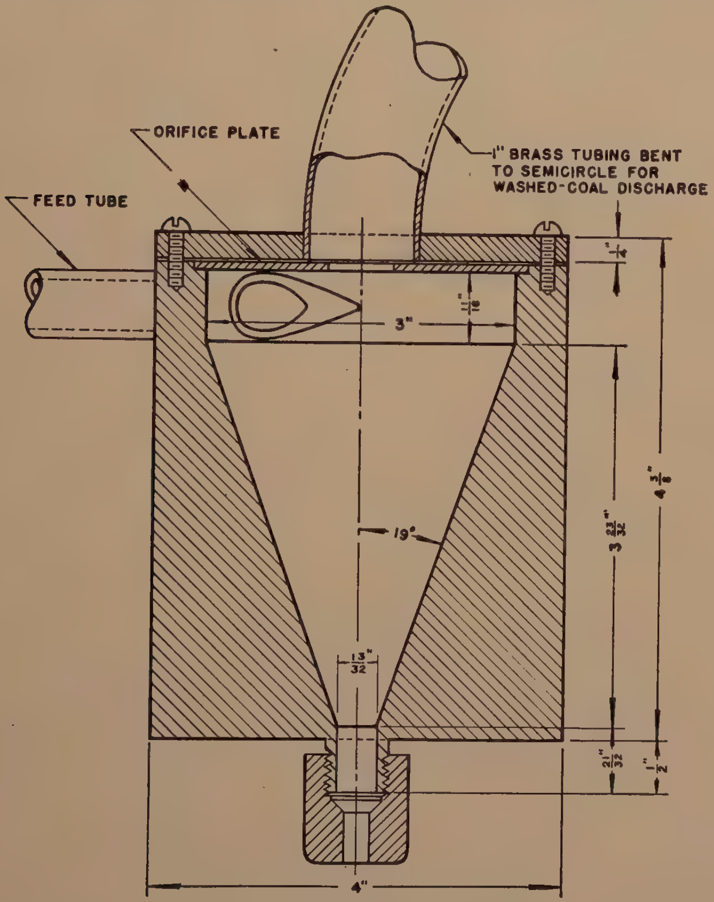

FIG 1—CONSTRUCTION OF CYCLONE.

In plant operation a centrifugal pump would be used to force the feed through the cyclone, but for batch tests in the laboratory a pressure tank is employed. A 20-lb sample of the feed coal is added to enough suspension to give a 6 to 1 pulp ratio in the pressure tank, where the mixture is kept from separating by a motor-driven agitator. Compressed air entering the tank at the bottom provides the pressure required to force the coal and suspension through the cyclone. In making a test, the tank is closed, the pressure brought up to the desired level, and the outlet valve opened. Refuse sprays from the bottom of the cyclone in a vortex, and washed coal also is forced through the orifice plate at the top of the cyclone in the form of a vortex. The washed coal and refuse produced during the first and last few seconds of the run are discarded, as they do not represent normal cyclone operation. The remainder of the washed coal and refuse is passed over screens to drain off the suspension and then rinsed with water. After drying, the products are weighed and sampled for float-and-sink examination and ash analysis.

The 3-in. cyclone, although it handles up to 800 lb of coal per hr, is smaller than desirable for testing purposes. However, so little was known about either the construction or operating characteristics of this device at the start of the investigation that it was imperative to use a small size, thus permitting a large number of tests to be made rapidly. The performance of a 3-in. cyclone may not duplicate that of a commercial unit, but the small cyclone should prove a satisfactory guide in predicting the influence of the various operating adjustments and should serve to demonstrate in a general way the fundamental characteristics of the separation between coal and refuse obtained in this device.

Coals Used

The coal employed in most of the tests was from the Bureau's Experimental mine at Bruceton, Pa., which is in the Pittsburgh bed. Most of the tests were made on coal sized between 14 and 35 mesh. Coarser material could not be treated in a small cyclone, and recovery of finer material from the suspension by screening is more difficult and time-consuming. A wider size range was used only in tests designed to demonstrate the performance of the cyclone on the finer sizes of coal.

As the Bruceton coal contains only about 5 pct of heavy impurity, and it was not desirable to make all tests on such a clean feed, sand of the same particle size as the coal was added to the feed for four tests. Sand is slightly heavier than the carbonaceous shale naturally associated with this coal, and of course the sand grains tend to be rounded rather than tabular. However, it is doubtful whether the substitution of sand for natural refuse material affected measurably the results of these tests.

Three cyclone tests were made on coal from the West No. 2 bed, Wilkeson, Washington. This coal is unusually difficult to wash because it contains both a high proportion of intermediate-density material and a large amount of heavy impurity.

Suspension

The heavy medium employed in all the tests described in this report was a suspension of barite ($BaSO_4$) in water, with bentonite added to increase stability. Both the barite and the bentonite were commercial products ground to pass 325-mesh. As prepared, a suspension of 1.40 sp gr contains 36.6 pct barite, 1 pct bentonite, and 62.4 pct water. Barite-bentonite suspensions have been used satisfactorily in the laboratory in the range from 1.20 to 1.44 sp gr. They are of acceptable viscosity and reasonably stable. The solids in the suspension settle out slowly and do not form a hard cake even on standing for long periods.

Barite was selected for the laboratory work because it happened to be available in finely pulverized form and because its physical properties made it suitable for use in the equipment available. Other media may be more suitable for use in a commercial plant because of either cost or physical characteristics. At the Netherlands State Mines, for example, flotation tailing from the fine-coal circuit in the plant has proved a suitable medium.[5] Research on the properties of heavy media suitable for use in the specific-gravity range employed in coal cleaning would be highly desirable, as the literature contains little information on this subject.

The "efficiency" of the separation between coal and impurity also is given. This efficiency is the ratio, expressed in per cent, of the yield of washed coal to the yield of float coal of the same ash content shown to be present in the feed by the specific-gravity analysis. These methods of evaluating washer performance are well known.

TABLE 1—*Analyses of Feed, Washed Coal, and Refuse from Cyclone Test No. 12*

Product	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative	
				Weight, Pct	Ash,[a] Pct
Washed coal weight, 95.7 pct	Under 1.30	64.7	2.8	64.7	2.8
	1.30 to 1.40	30.2	7.5	94.9	4.3
	1.40 to 1.50	3.4	18.3	98.3	4.8
	1.50 to 1.60	1.4	27.2	99.7	5.1
	1.60 to 1.70	0.3	31.8	100.0	5.2
	Over 1.70	0.0		100.0	5.2
Refuse, weight, 4.3 pct	Under 1.30	0.2	2.2	0.2	2.2
	1.30 to 1.40	0.2	8.0	0.4	5.1
	1.40 to 1.50	0.2	20.6	0.6	10.3
	1.50 to 1.60	2.8	30.6	3.4	27.0
	1.60 to 1.70	10.5	37.9	13.9	35.2
	Over 1.70	86.1	67.8	100.0	63.3
Feed, calculated from products	Under 1.30	61.9	2.8	61.9	2.8
	1.30 to 1.40	28.9	7.5	90.8	4.3
	1.40 to 1.50	3.3	18.3	94.1	4.8
	1.50 to 1.60	1.5	27.5	95.6	5.1
	1.60 to 1.70	0.7	35.5	96.3	5.4
	Over 1.70	3.7	67.8	100.0	7.7

CALCULATED DISTRIBUTION OF SPECIFIC-GRAVITY FRACTIONS

Specific Gravity	Percentage Sent to Washed Coal	Percentage Rejected as Refuse
Under 1.30	100.0	0.0
1.30–1.40	100.0	0.0
1.40–1.50	99.7	0.3
1.50–1.60	91.8	8.2
1.60–1.70	39.2	60.8
Over 1.70	0.0	100.0

[a] Moisture-free basis.

METHOD OF EXPRESSING RESULTS

The washed coal and refuse from all the cyclone tests were examined by means of float-and-sink and ash analyses. Wherever possible in this report the results of the cyclone tests are presented in conventional manner—that is, they are evaluated by the ash contents of the products, and the percentage of sink material in the washed coal and float material in the refuse at an appropriate specific gravity.

Certain features of cyclone operation can be demonstrated best, however, by means of distribution curves of the type developed in this country by the Bureau of Mines[6] and in the Netherlands by K. F. Tromp.[7] The distribution curve shows what percentage of each specific gravity fraction of the feed was sent to the washed coal and what was rejected as refuse. As an illustration of the distribution-curve method of expressing washing results, Table 1 gives specific-gravity analyses of the feed, washed coal, and refuse from a cyclone test on 14 to 35-mesh Bruceton coal, and, in the lower part, the calculated distribution of the specific-gravity fractions between washed coal and refuse. The distribution curve itself is plotted in Fig 2.

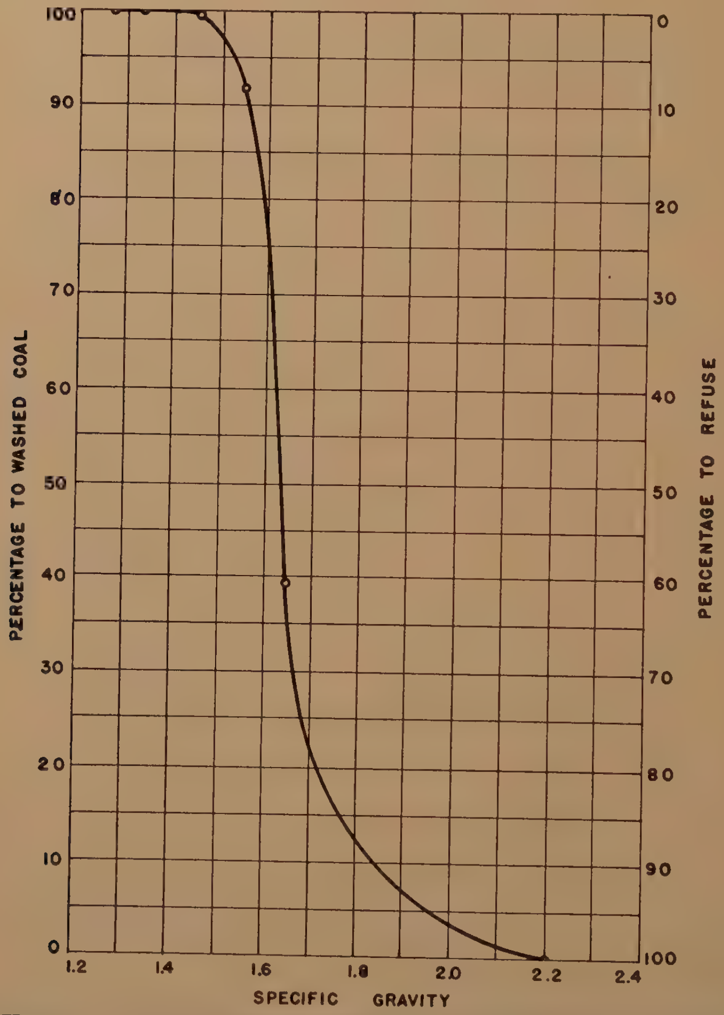

Fig 2—Distribution of feed between washed coal and refuse for cyclone test No. 12.

The distribution data for each specific-gravity fraction are plotted against the mean specific gravity of the fraction, mean specific gravity being defined as the specific gravity on which half of the material would float and half would sink. An assumption is involved for the material lighter than the lowest specific gravity and the material heavier than the highest specific gravity used in the float-and-sink test. For Bruceton coal, it was assumed that no coal was lighter than 1.25 sp gr and no refuse was heavier than 2.60 sp gr. The distribution data for float 1.30 coal are plotted therefore at 1.27 sp gr and the data for sink 1.70 or 1.80 material are plotted at 2.20 sp gr. Any error involved in making these assumptions for the lightest and heaviest material has relatively little influence on the shape of the distribution curve.

Fig 2 indicates that none of the low-gravity coal was lost in the refuse product, and none of the impurity heavier than 1.70 sp gr entered the washed coal. The curve crosses the 50 pct weight abscissa at 1.65 sp gr and this, by definition, is the specific gravity of separation. Material of this specific gravity was divided equally by the cyclone between washed coal and refuse. A distribution curve representing perfect separation would show all coal lighter than the specific gravity of separation going to washed coal and all heavier material going to refuse, with the curve passing from 100 pct to 0 on the ordinate representing the specific gravity of separation. The degree to which the distribution curve approximates this theoretical or perfect shape is a measure of the accuracy or sharpness of the separation made by the washer.

Driessen,[5] who calls the distribution curve an "error" curve, terms the area between the distribution curve and the theoretical curve the "error area." Expressed in square centimeters when the curve is plotted to a definite scale, the error area serves as a single figure characterizing the sharpness of the separation effected between clean coal and refuse. The principal objection to the error-area method of evaluating washer performance is the fact that the inclusion of a few tenths of one per cent of impurity in the washed coal increases the error area more than the loss of several per cent of clean coal in the refuse. With most separations the inclusion of a fraction of one per cent of impurity in the washed coal is not serious, but the loss of clean coal in the refuse product is a matter of much greater concern. Nevertheless, the distribution curve and error area are useful tools in demonstrating certain features of washer performance that cannot be shown readily by ash analyses or an efficiency value.

INFLUENCE OF WASHED-COAL AND REFUSE OPENINGS ON PERFORMANCE OF CYCLONE

Table 2 summarizes the results of a series of cyclone tests made to illustrate the influence of the size of the washed-coal orifice on the performance of the cyclone. These tests were made on 14 to 35-mesh Bruceton coal. The size of the feed and refuse openings, the operating pressure, and the specific gravity of the medium were held constant, while the washed-coal orifice was varied from 5/16 to 3/4 in. Fig 3 shows distribution curves for the four tests.

TABLE 2—*Summary of Cyclone Tests Showing Influence of Size of Washed-coal Orifice*[a]

Data	Test No.			
	27	24	25	26
Washed-coal orifice, in.	5/16	7/16	9/16	3/4
Capacity, ton per hour	0.26	0.28	0.32	0.41
Sink 1.60 sp gr in feed, pct	4.0	4.5	3.9	3.6
Yield of refuse, pct	28.6	4.6	2.6	2.1
Ash in feed, pct	7.1	7.6	7.2	7.1
Ash in washed coal, pct	5.3	5.0	5.5	5.6
Ash in refuse, pct	11.5	59.6	71.2	75.6
Specific gravity of separation	1.60	1.60	1.80	2.00
Sink 1.60 sp gr in washed coal, pct	0.3	0.3	1.4	1.6
Float 1.60 sp gr in refuse, pct	86.8	9.8	0.5	3.3
Efficiency, pct	73.3	99.9	100.0	100.0

[a] Tests made with Bruceton coal, 14 to 35-mesh; barite suspension of 1.40 sp gr; feed-tube opening 7/16 in., refuse opening 13/32 in.; 20 psi pressure.

Test 24, made with a washed-coal orifice of 7/16 in., represents the best operation in the series. The feed contained 4.5 pct of material heavier than 1.60 sp gr, and the cyclone rejected 4.6 pct of refuse. The washed coal contained 0.3 pct of material heavier than 1.60 sp gr, and the refuse contained 9.8 pct of float coal. The distribution curve for this test shows that only 0.1 pct of the coal lighter than 1.30 sp gr and 0.2 pct of the coal between 1.30 and 1.40 sp gr was lost in the refuse. Of the material heavier than 1.80 sp gr, 98.5 pct was rejected as refuse and only 1.5 pct entered the washed coal.

In tests 25 and 26, successively larger washed-coal orifices were used, and as a result the cyclone produced too little refuse in these tests. When the washed-coal orifice is too large—or, conversely, when the refuse opening is too small—the cyclone makes a separation at a higher specific gravity, and heavier material enters the washed coal. For example, in test 26 the specific gravity of separation was 2.00, and the washed coal contained 1.6 pct of material heavier than 1.60 sp gr. Little coal is lost in the refuse under these

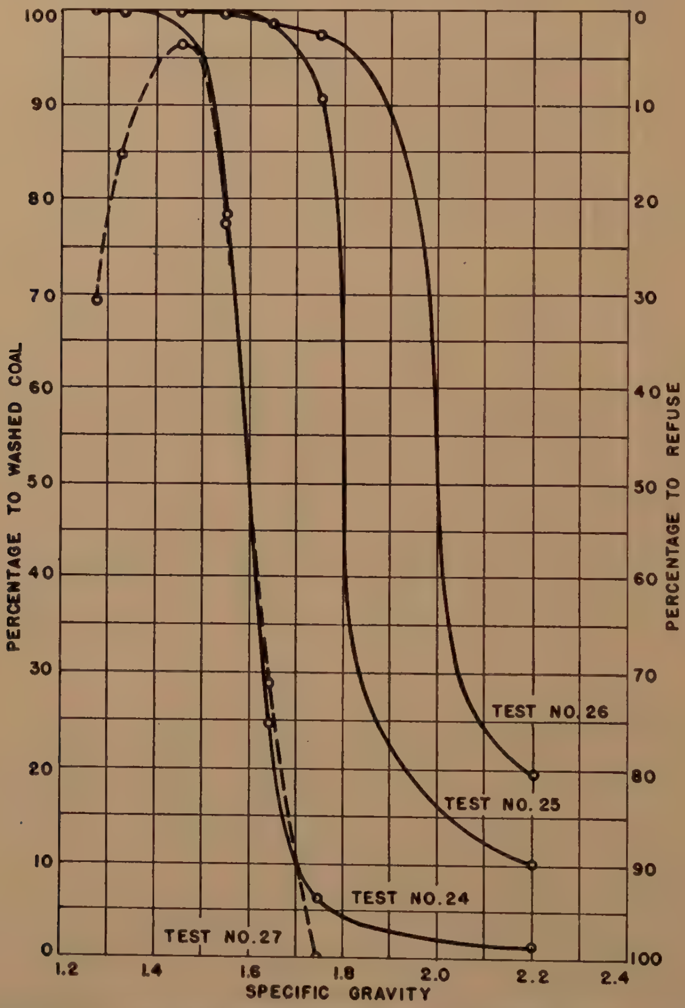

Fig 3—Distribution of feed between washed coal and refuse, illustrating influence of size of washed-coal orifice.

conditions and consequently the efficiency of the operation is high.

Test 27 represents the other extreme in cyclone operation, for the washed-coal orifice was too small. If the washed-coal orifice is too small or the refuse opening too large, too much material is removed as refuse, and there is a loss of the cleanest coal present in the feed. The distribution curves in Fig 3 illustrate clearly that with an undersize washed-coal opening it is the cleanest coal rather than the material of intermediate specific gravity that is lost in the refuse product. The treatment of intermediate-density material, and hence the specific gravity of separation, was almost identical in tests 24 and 27. In this respect the cyclone differs from other types of washers; with conventional washing units, withdrawing too much refuse does not give a distribution curve exhibiting the "hook" shown for test 27 in Fig 3.

TABLE 3—*Summary of Cyclone Tests Showing Influence of Washed-coal and Refuse Openings*[a]

Data	Test No.					
	7	10	12	8	11	9
Washed-coal orifice, in	7/16	5/16	5/16	7/16	5/16	5/16
Refuse opening, in	3/16	3/16	7/32	5/16	¼	5/16
Area of refuse opening, percentage of washed-coal orifice	42.9	60.0	70.0	71.4	80.0	100.0
Sink 1.60 sp gr in feed, pct		4.6	4.4	5.4	4.7	4.7
Yield of refuse, pct	2.9	3.6	4.3	4.4	8.8	23.3
Ash in feed, pct		7.6	7.7	8.1	7.1	7.7
Ash in washed coal, pct		5.4	5.2	5.5	4.8	5.7
Ash in refuse, pct		64.8	63.3	63.5	31.0	14.7
Specific gravity of separation	2.14	1.87	1.64	1.65	1.63	1.65
Sink 1.60 sp gr in washed coal, pct	2.0	1.3	0.3	0.5	0.4	0.6
Float 1.6p sp gr in refuse, pct	0.2	1.0	3.4	5.3	51.3	81.8
Efficiency, pct		99.9	99.9	99.9	95.4	79.1

[a] Tests made on 14 to 35-mesh Bruceton coal; feed-tube opening, ¼ in.; barite suspension of 1.44 sp gr; operating pressure, 20 psi.

It is noteworthy that a small decrease in the size of the washed-coal orifice has a much greater effect than a corresponding increase. Decreasing the washed-coal orifice from 7/16 to 5/16 in. caused the amount of refuse to jump from 4.6 to 28.6 pct, while increasing the orifice from 7/16 to ¾ in., a much greater change in size, decreased the amount of refuse only from 4.6 to 2.1 pct.

The data in Table 2 demonstrate that the size of the washed-coal opening, in addition to affecting the yield of washed coal, influences the capacity of the cyclone. In increasing the washed-coal orifice from 5/16 to ¾ in., a ratio of 5.8 to 1 in area, the capacity of the cyclone increased from 0.26 to 0.41 ton per hr, an increase of 58 pct. Other tests have shown that in treating a coal containing only a small amount of impurity the size of the refuse opening has no measurable influence on the capacity of the cyclone. Whether this would be true when a large amount of refuse is removed is not known. The phenomenal capacity of the cyclone is realized when it is borne in mind that 0.4 ton per hr was obtained with a device only 3 in. in diameter. At the Netherlands State Mines a pilot-plant cyclone 16 in. in diameter treats 15 tons per hr.

Table 3 summarizes the results of a series of six cyclone tests made on 14 to 35-mesh Bruceton coal using a suspension of 1.44 sp gr. In these tests the feed opening remained constant and the washed-coal and refuse openings were varied to give successively larger percentages of refuse. Like the data shown in Table 2, these tests demonstrate that either an overly large washed-coal opening or an undersize refuse opening causes the separation between coal and impurity to be made at a higher specific gravity. Normal operation of the cyclone with a suspension of 1.44 sp gr gives a separation between coal and impurity at about 1.65 sp gr. Thus the centrifugal force developed in the cyclone adds about 0.2 to the specific gravity of the medium. This increase in specific gravity is the same as that shown by the tests in Table 2, where a medium of 1.40 sp gr gave a separation at about 1.60 specific gravity.

INFLUENCE OF FEED-TUBE OPENING ON PERFORMANCE OF CYCLONE

As the size of the opening through which the feed enters the cyclone was thought to be a factor in determining capacity, a series of three tests was conducted in which the feed opening was varied from ¼ to 7/16 in. These tests were made on 14 on 35-mesh Bruceton coal, using a suspension of 1.40 sp gr, while all factors other than feed-tube opening were held constant. The data in Table 4 show that only 9 pct increase in capacity, from 0.22 to 0.27 ton per hr, was obtained when the feed-tube opening was increased by 300 pct in area. Thus the feed opening has much less influence than the washed-coal orifice in determining the capacity of the cyclone. If the washed-coal orifice were grossly oversized, the feed opening might conceivably control capacity, but of course under these conditions little if any cleaning action would be obtained. As neither capacity nor cleaning performance was affected appreciably in these three tests, the feed opening can be considered a factor of little importance in cyclone operation.

TABLE 4—*Summary of Cyclone Tests Showing Influence of Feed-tube Opening*[a]

	Test No.		
	22	21	20
Size of feed tube, in	¼	5/16	7/16
Capacity, ton per hour	0.22	0.25	0.27
Sink 1.60 sp gr in feed, pct	4.7	5.1	5.0
Yield of refuse, pct	3.8	3.8	3.4
Ash in feed, pct	8.1	8.4	7.9
Ash in washed coal, pct	5.7	5.9	5.7
Ash in refuse, pct	70.6	71.5	72.0
Specific gravity of separation	1.75	1.77	1.77
Sink 1.60 sp gr in washed coal, pct	1.4	1.4	1.3
Float 1.60 sp gr in refuse, pct	0.5	0.3	0.5
Efficiency, pct	99.8	100.0	99.9

[a] Tests made with Bruceton coal, 14 to 35-mesh; barite suspension of 1.40 sp gr; washed-coal orifice 7/16 in., refuse opening 5/16 in.; operating pressure 20 psi.

INFLUENCE OF OPERATING PRESSURE ON PERFORMANCE OF CYCLONE

Table 5 summarizes the results of three tests conducted to determine the influence of the amount of air pressure used to force the feed through the cyclone. These tests were made with 14 to 35-mesh Bruceton coal also, using a barite suspension of 1.40 sp gr, and all factors other than operating pressure were held constant. Capacity of the cyclone decreased with decrease in pressure, the relationship between the two being almost a straight-line function in the range of 10 to 20 psi investigated. As the pressure decreased, other variables remaining constant, the specific gravity of the separation between coal and impurity remained the same, but more light coal entered the refuse product and consequently the efficiency of the separation was impaired. As the results obtained at 20 psi were superior to those obtained at lower pressures, tests at pressures higher than the 20 psi would be desirable. However, operating at higher pressures was not feasible with the equipment then in use. Tests at higher pressures will be made in the future.

The operating pressure reported is measured in the pressure tank rather than at the cyclone The actual pressure at the feed opening of the cyclone is not known, but probably is considerably less than that measured in the tank.

TABLE 5—*Summary of Cyclone Tests Showing Influence of Operating Pressure*[a]

Data	Test No.		
	24	28	29
Operating pressure, psi	20	15	10
Capacity, ton per hour	0.28	0.24	0.19
Sink 1.60 sp gr in feed, pct	4.5	3.8	2.9
Yield of refuse, pct	4.6	4.2	3.6
Ash in feed, pct	7.6	7.2	6.4
Ash in washed coal, pct	5.0	4.9	4.7
Ash in refuse, pct	59.6	58.6	54.0
Specific gravity of separation	1.60	1.60	1.60
Sink 1.60 sp gr in washed coal, pct	0.3	0.2	0.1
Float 1.60 sp gr in refuse, pct	9.8	12.2	21.2
Efficiency, pct	99.9	100.0	99.3

[a] Tests made with 14 to 35-mesh Bruceton coal; barite suspension of 1.40 sp gr; feed-tube opening 7/16 in.; washed-coal orifice 7/16 in.; refuse opening 13/32 in.

INFLUENCE OF VARYING AMOUNT OF IMPURITY IN FEED ON CYCLONE OPERATION

The feed to any coal washer varies considerably in quality, hence it was desirable to determine how well the cyclone could cope with variations in the amount of impurity in the feed. Mixtures of Bruceton coal and sand, 14 to 35-mesh in size, ranging from 10.0 to 24.6 pct in ash content and containing 6.2 to 21.9 pct of impurity heavier than 1.60 sp gr, were fed to the cyclone without making any compensating changes in the size of the washed-coal and refuse openings. Results of these tests are summarized in Table 6.

The clean coal produced was of substantially uniform quality; with the cleanest feed the washed coal contained 4.8 pct ash and 0.1 pct of impurity heavier than 1.60 sp gr, while with the dirtiest feed the washed coal contained 4.9 pct ash and 0.3 pct of impurity heavier than 1.60 sp gr. The amount of coal lost in the refuse product was greater with the cleaner feeds, but even with the cleanest feed the 15.6 pct of coal lost in the refuse is not excessive.

The degree to which the cyclone adjusts itself to changes in quality of feed is seen best in Table 6, by comparing the amount of impurity present with the corresponding amount of refuse removed and with the amount of impurity in the refuse expressed as a percentage of the feed. The amount of refuse removed varies directly with the amount of impurity present, the ratio between the amount of impurity in the refuse and that in the feed remaining substantially constant; in test 33 the refuse contained 98 pct of the heavy impurity in the feed, and in test 35 it contained 99 pct.

TABLE 6—*Summary of Cyclone Tests Showing Influence of Variations in Amount of Impurity in Feed*[a]

Data	Test No.			
	33	32	34	35
Sink 1.60 sp gr in feed, pct	6.2	10.3	13.1	21.9
Yield of refuse, pct	7.2	11.0	13.7	22.3
Sink 1.60 sp gr in refuse, pct of feed	6.1	10.0	13.0	21.7
Sink 1.60 sp gr in washed coal, pct	0.1	0.2	0.2	0.3
Float 1.60 sp gr in refuse, pct	15.6	9.0	5.5	2.8
Ash in feed, pct	10.0	13.3	16.2	24.6
Ash in washed coal, pct	4.8	4.8	4.9	4.9
Ash in refuse, pct	76.5	81.1	87.5	92.9
Efficiency, pct	98.9	99.7	99.3	99.2

[a] Tests made on mixtures of Bruceton and sand, 14 to 35-mesh size; barite suspension of 1.40 sp gr; feed-tube opening 7/16 in.; washed-coal orifice 3/8 in.; refuse opening 3/8 in.; operating pressure 20 psi.

Considering how critical the size of the washed-coal and refuse openings are, as demonstrated by the data in Tables 2 and 3, the ability of the cyclone to cope with radical changes in quality of feed is quite surprising.

TEST ON 14 TO 100-MESH BRUCETON COAL

One of the principal limitations of heavy-medum processes is their inability to deal with fine material. Although some information in the literature indicates the use of heavy-medium separation on material as fine as 10 mesh, the lower limit of size treated in plants now operating is much larger. In order to investigate the ability of the cyclone cleaner to deal with material finer than the 14 to 35-mesh coal cleaned in the tests previously described, a test was made on a wider range of sizes. Coal 14-mesh to 0 was fed to the cyclone. The washed coal and refuse were recovered on a 100-mesh screen; recovery of the finer coal from the suspension was not feasible, although it could have been done by flotation. Table 7 summarizes the washing results by individual size fractions, and Fig 4 shows distribution curves for the various sizes.

TABLE 7—*Summary of Results Obtained in Cyclone Test No.* 36 *on* 14 *to* 100-*mesh Bruceton Coal*[a]

Data	Screen Size, Mesh					
	14–35	35–48	48–65	65–100	–100	14–100
Screen analysis of feed, pct	66.3	10.2	7.5	4.9	11.1	88.9
Sink 1.60 sp gr in feed, pct	4.6	4.6	4.6	5.5		4.7
Yield of refuse, pct	4.9	5.0	6.5	9.0		5.3
Specific gravity of separation	1.60	1.67	1.68	1.88		
Sink 1.60 sp gr in washed coal, pct	0.2	0.9	1.2	2.4		0.5
Float 1.60 sp gr in refuse, pct	9.6	25.7	47.2	62.1		20.8
Ash in feed, pct	7.4	6.8	6.8	7.4	6.5	7.3
Ash in washed coal, pct	4.8	4.6	4.8	5.6		4.9
Ash in refuse, pct	57.8	48.0	35.0	26.2		51.0
Efficiency, pct	99.7	98.8	96.8	93.6		99.0

[a] Test made on 14-mesh to 0 Bruceton coal; barite suspension of 1.40 sp gr; feed-tube opening 7/16 in.; washed-coal orifice 7/16 in.; refuse opening 13/32 in.; operating pressure 20 psi.

The finer sizes of coal were separated at higher specific gravities than the 14 to 35-mesh material; the 65 to 100-mesh fraction was separated at 1.88 sp gr in comparison with the separation at 1.60 sp gr obtained in the 14 to 35-mesh size. The amount of heavy impurity remaining in the washed coal increased with decreasing particle size, ranging from 0.2 pct in the 14 to 35-mesh size to 2.4 pct in the 65 to 100-mesh material. Similarly, the amount of coal lost in the refuse product increased with decreasing particle size, ranging from 9.6 pct in the 14 to 35-mesh size to 62.1 pct in the 65 to 100-mesh fraction. Thus the efficiency of the separation between coal and impurity decreased with decrease in particle size. Only the material coarser than about 48-mesh was handled with satisfactory efficiency. Below that size the sharpness of the separation dropped rapidly.

An interesting feature of this test is the fact that the separation obtained on the 14- to 35-mesh fraction of the 14- to 100-mesh feed was almost identical with that obtained in test 24, which was made with the same cyclone adjustments but with a feed sized 14- to 35-mesh. In other words, the presence of material finer than 35-mesh did not alter the operation of the cyclone on the coarser sizes.

The fact that any cleaning device operating on an unsized feed is less efficient in treating the finer sizes is well known, and yet, except for pneumatic cleaners,[8,9] the literature contains almost no quantitative data on the degree to which efficiency is impaired in the fine sizes. Fig 5 is of interest therefore because it compares the relationship between efficiency and particle size obtained with the cyclone cleaner with that representing the operation of a coal-washing table. Data for the coal-washing table were obtained from a test on Colorado coal, partial results of which were reported by Bird and Yancey.[10] As this table test was exceptionally efficient, it provides a critical comparison for the cyclone cleaner. However, Fig 5 should not be construed as a comparison of the over-all operation of the cyclone and table, for the two coals involved have different washability characteristics. What is intended is merely to show that the relationship between particle size and cleaning efficiency obtained with the cyclone is similar to that characterizing table operation. With both types of cleaners, efficiency drops off sharply in the range between 48- and 65-mesh.

The inability of the cyclone to clean the finer sizes with the same high efficiency exhibited on material coarser than 35-mesh probably is related to the viscosity of the medium employed. With a more viscous medium the separation obtained in the finer sizes probably would be even poorer than that shown in this test, while a medium of less viscosity than the barite-bentonite suspension employed might give somewhat more favorable results on fine

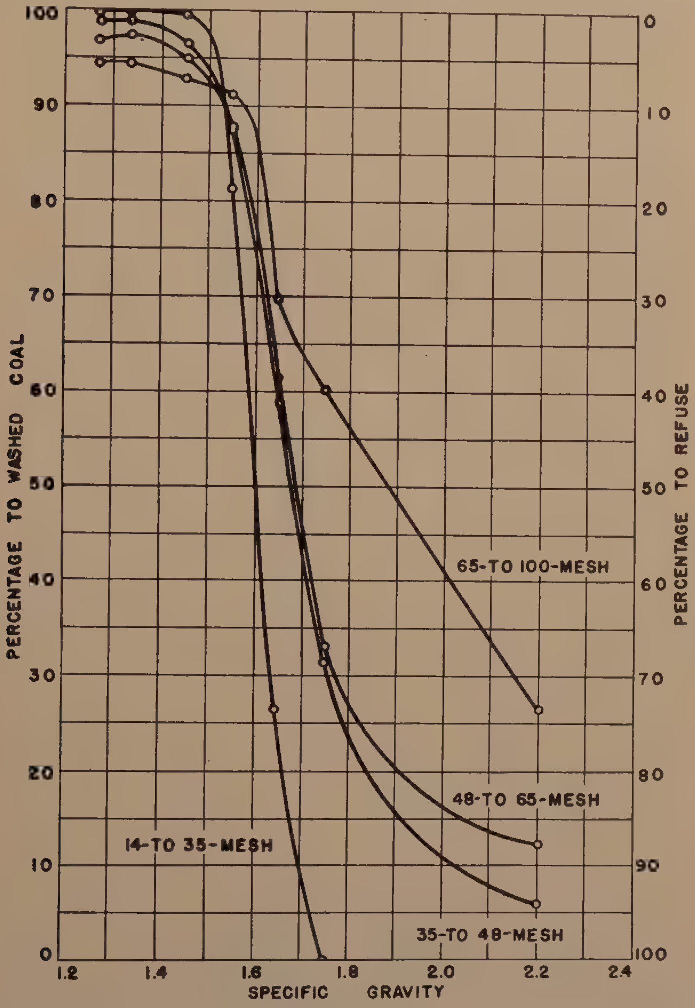

FIG 4—DISTRIBUTION OF FEED BETWEEN WASHED COAL AND REFUSE, ILLUSTRATING INFLUENCE OF PARTICLE SIZE.

material. However, even though the cyclone is unable to make a "float-and-sink" separation on fine coal down to zero size, it does extend greatly the range of sizes amenable to treatment by existing heavy-medium processes.

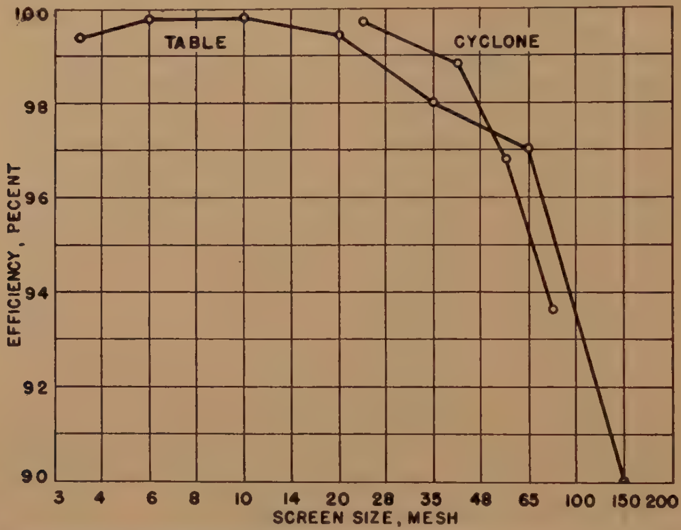

FIG 5—RELATIONSHIP BETWEEN EFFICIENCY AND PARTICLE SIZE FOR CYCLONE AND COAL-WASHING TABLE.

Treatment of all sizes down to zero is of course, a desirable operating characteristic for a heavy-medium process. However, treatment of the extreme fines necessarily involves recovery of the fine material to prevent contamination of the medium. Unless a magnetic medium is employed, flotation provides the only means of separating the fine sizes of coal from the suspension. The difficulties involved in the separation of fine coal from cleaning media may well outweigh the advantages in cleaning performance that might be obtained in treating the extreme fines by heavy-medium separation. However, this aspect of the problem is beyond the scope of the present report.

TESTS WITH WILKESON COAL

In order to investigate the performance of the cyclone on a coal more difficult to wash than Bruceton, a series of tests was made with 14 to 35-mesh coal from the west dip of the No. 2 bed, Wilkeson, Washington. This coal is unusually difficult to wash, for it contains about 35 pct of impurity heavier than 1.60 sp gr and only about 50 pct of clean coal lighter than 1.40. Table 8 and Fig 6 show the results of these tests, all of which were made with a suspension of 1.30 sp gr, the washed coal and refuse openings being varied to secure different yields of washed coal.

TABLE 8—*Summary of Results of Cyclone Tests with Wilkeson Coal*[a]

Data	Test No.		
	39	42	41
Washed-coal orifice, in.	7/16	1/2	9/16
Refuse opening, in.	13/32	13/32	3/8
Yield of washed coal, pct	39.9	51.9	60.1
Ash in feed, pct	31.0	30.9	30.2
Ash in washed coal, pct	8.4	10.2	11.5
Ash in refuse, pct	46.0	53.2	58.6
Specific gravity of separation	1.37	1.44	1.50
Efficiency, pct	92.4	92.5	96.6

[a] Tests made on 14 to 35-mesh Wilkeson coal; barite suspension, 1.30 sp gr; feed-tube opening 7/16 in.; operating pressure, 20 psi.

The 0.2 difference between the specific gravity of the suspension and the specific gravity of separation obtained in previous tests with Bruceton coal was obtained with Wilkeson coal only in test No. 41. Tests 39 and 42, in which more refuse was removed, showed separations at 1.37 and 1.44, respectively. Thus, with the Wilkeson coal, withdrawing too much refuse decreased the specific gravity of separation, an effect not observed with the Bruceton coal. This difference in the performance of the cyclone with these two coals probably is related to the much higher proportion of intermediate-density material in the Wilkeson coal.

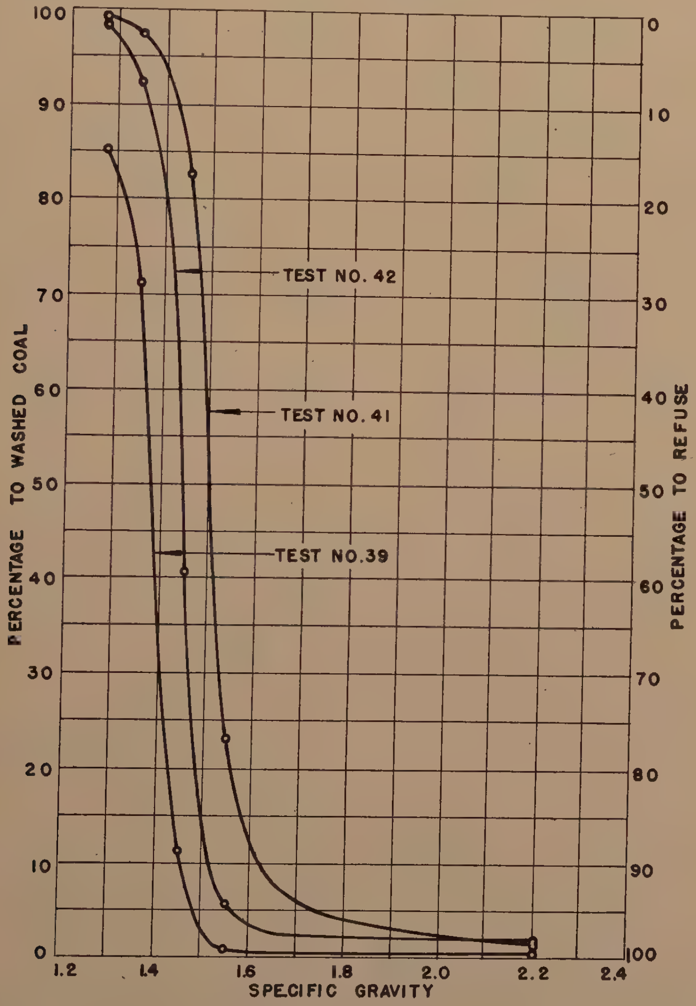

FIG 6—DISTRIBUTION OF FEED BETWEEN WASHED COAL AND REFUSE, FOR TESTS MADE WITH WILKESON COAL.

The cyclone produced washed coal ranging from 8.4 to 11.5 pct ash from feed averaging 30.7 pct, at yields of 40 to 60 pct. The efficiency of the separation ranged from 92.4 pct with the cleanest washed coal to 96.6 pct with the washed coal of

11.5 pct ash. These efficiencies are much lower than those obtained with Bruceton coal, and, like the distribution curves in Fig 6, they demonstrate that the cyclone was unable to make as sharp a separation between coal and impurity with the Wilkeson coal as with the more amenable Bruceton coal. This was expected, of course, because a coal containing so much heavy impurity and so much intermediate-density material cannot be cleaned by any method with the high efficiency obtainable in treating coals that are easier to wash.

The efficiency obtained with the cyclone in treating such a dirty feed is difficult to interpret except in terms of comparison with a cleaning process of known characteristics. Consequently, a comparative test was made on 14 to 35-mesh Wilkeson coal, using a laboratory-size Wilfley table. Zonal samples were taken from the discharge of the table, so that washed coal of various ash contents could be obtained. Fig 7 compares the performance of the cyclone with that of the table by means of the familiar yield-ash curve; the yields of washed coal for the cyclone were adjusted to correspond to a cyclone feed of the same ash content as that of the table feed.

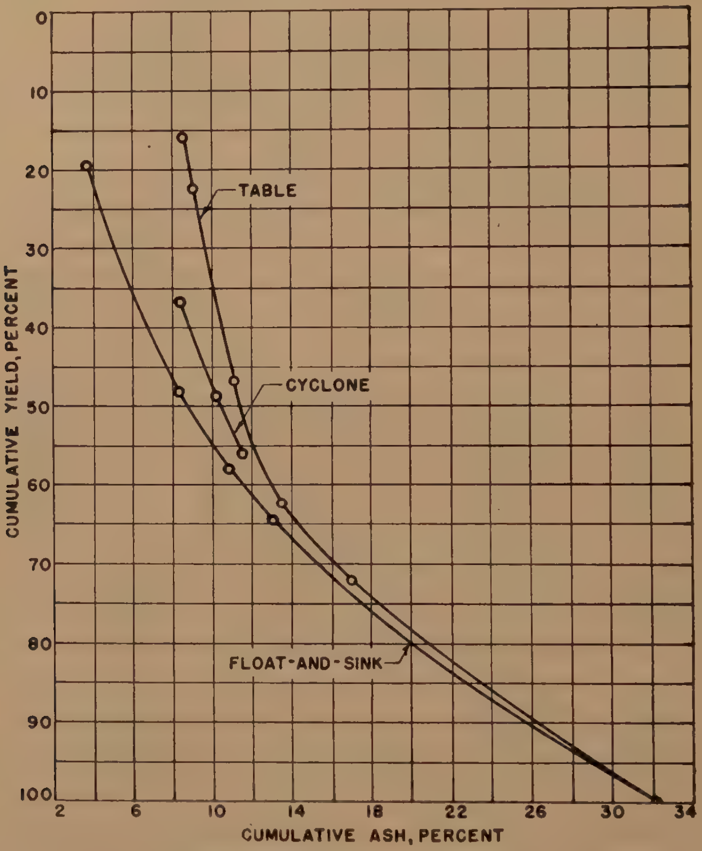

FIG 7—YIELD-ASH CURVES REPRESENTING CYCLONE AND TABLE OPERATION.

At 8.5 pct ash, the cleanest product made by the table, the yield was 16.0 pct, while the yield obtained with the cyclone was 36.7 pct. The cyclone was superior to the table at all ash contents of washed coal, but the margin of superiority was greatest at the lower ash contents. At high ash contents the performance of cyclone and table probably would be substantially the same. This comparison thus illustrates that the natural field of application for the cyclone cleaner is on difficult materials where its inherently sharp separation between coal and impurity is of greatest importance. With coals that are easy to clean almost any cleaning process will give a product of the required ash content at high efficiency, but with coals that are more difficult to clean, in the re-treatment of middling material, or in making separations at low specific gravity on any coal, only a cleaning process of inherently high efficiency is suitable.

SUMMARY

This report describes an investigation made by the Bureau of Mines of a new heavy-medium coal-cleaning device under development in the Netherlands. Preliminary tests of a laboratory-size unit—known as the cyclone coal washer—have demonstrated its high capacity and its ability to make a sharp, efficient separation between coal and impurity at particle sizes down to about 48-mesh. Thus, the cyclone offers promise of extending substantially the size range now treated by heavy-media processes. The principal factors affecting operation of the cyclone are the specific gravity of the medium employed and the size of the openings through which the clean coal and refuse are discharged.

ACKNOWLEDGMENTS

This investigation was conducted by the Bureau of Mines in cooperation with the College of Mines of the University of Washington; the authors are indebted to Milnor Roberts, dean, and Joseph Daniels, professor of mining engineering and metallurgy, for continuing support. The work was done under the general direction of A. C. Fieldner, Chief Fuels and Explosives Branch, Bureau of Mines.

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DISCUSSION

M. G. DRIESSEN*—Before making any comments on this excellent and systematic piece of work, I wish to express my admiration for the method both authors have adopted to investigate the characteristics of the cyclone washer and for the short lapse of time they required to obtain complete data on a hitherto unknown coal cleaner. The general arrangement of this type of washer requires a different method of operation, a pressure vessel, and changing of characteristic items in the cyclone necessary to obtain comparable results. Moreover, the time for each test is only a few minutes, or sometimes even less than one minute, and this entails a different way of discussing the results than with the usual washers. All this new experience had to be acquired by the authors, and it is almost amazing that within a few months they are able to give such a comprehensive report about the subject.

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The remarks I am going to make should, therefore, not be considered as criticism but as a basis of discussion for different viewpoints.

As a matter of minor importance it would be advisable to define in an exact manner expressions such as "efficiency" and "recovery." Geer and Yancey clearly state on p. 223 the definition of "efficiency" but in some publications the word "recovery" is used.

On p. 225 the authors object to the use of "error area." The remark is quite to the point, especially in cases where the refuse constitutes only a small fraction (3 to 4 pct) of the feed. As the error curve is based on the part of a specific gravity fraction that goes the wrong way, this part, *expressed in percentage of the specific gravity fraction,* may be relatively large, whereas the fraction itself is small if the total refuse is only a few percentages. Therefore, a better indication of the error area would be to give two figures; for instance 10 to 15. This would express the part of the error area as indicating the error in the refuse is 10 units and the part indicating the error in the coal is 15 units.

Geer and Yancey's paper does not give any error area, and it would be well to have them in order to be able to compare the error curves among themselves as well as with other publications. In this way Geer and Yancey's data could be better evaluated.

The assumption on p. 224 of the paper that no coal lighter than 1.25 would be present in the feed seems to me logical, and this would not interfere with a correct plotting of the error curve, especially as the lightest specific gravity used for analyzing the washed coal and refuse is very close to 1.25. The other assumption—that no refuse is heavier than 2.6—is also approximately true, but plotting the part heavier than 1.7 at an intermediate specific gravity of 2.2 seems to me somewhat arbitrary and would affect considerably the shape, and therefore the area, of the error curve. This applies for the error curves in Fig 2, 3, 4 and 6.

Although the authors state clearly on p. 230 that efficiencies of cyclone and coal-washing tables cannot be compared because the washability of two different coals will always be involved, Fig 5 stays longer in one's memory than the reading of a report and must be considered to be misleading in that efficiencies can be compared only if exactly the same feed and separation *at exactly the same* specific gravity are implied.

As an example of how the efficiency changes with specific gravity of separation, the following table, calculated for coal from the Saxton mine, Indiana, may be of interest:

Specific Gravity of Separation	Efficiency (or Recovery), Pct
1.26	42.5
1.27	56.5
1.30	89.0
1.35	99.0
1.40	99.7

The calculation of the efficiencies is based on the same error curve for heavy-medium separation at all of the specific gravities mentioned above; so, instead of plotting the efficiencies, it would be advisable to plot the error areas of the different size fractions as functions of the screen sizes.

In comparing the results of the "difficult" Wilkeson coal with the "easy" Bruceton coal, the authors state on p. 234 that the efficiencies of separation of the first are "much lower than those obtained with Bruceton coal, and like the distribution curves shown in Fig 6, they demonstrate that the *cyclone was unable to make as sharp a separation* between coal and impurity with the Wilkeson coal as with the more amenable Bruceton coal."

Here again efficiencies are compared where this is not allowable, as the Wilkeson coal was separated at low specific gravities (1.37, 1.44 and 1.50) whereas most of the Bruceton coal (as is shown in the paper) was separated at a much higher specific gravity and because the washability curves differ. Moreover, the distribution curves or error curves of Fig 6 (Wilkeson coal) are not worse than those of Fig 3 (Bruceton coal). The error area of test No. 42 of Fig 6 is 23 sq cm if the correction for the specific gravity larger than 1.6 is taken into account (see previous comment) and if the same units are assumed, as in the paper No. 5 of the references, while the error curve of test No. 24 of Fig 3 is 28 sq cm. This emphasizes the use of the error areas. Whereas the efficiencies may change considerably with another feed and another specific gravity of separation, the shape of the error curves and the error areas remains practically the same as long as the cyclone is adjusted properly; and the major variable, of course, is the particle size as shown in Geer and Yancey's paper and in Ref. 5.

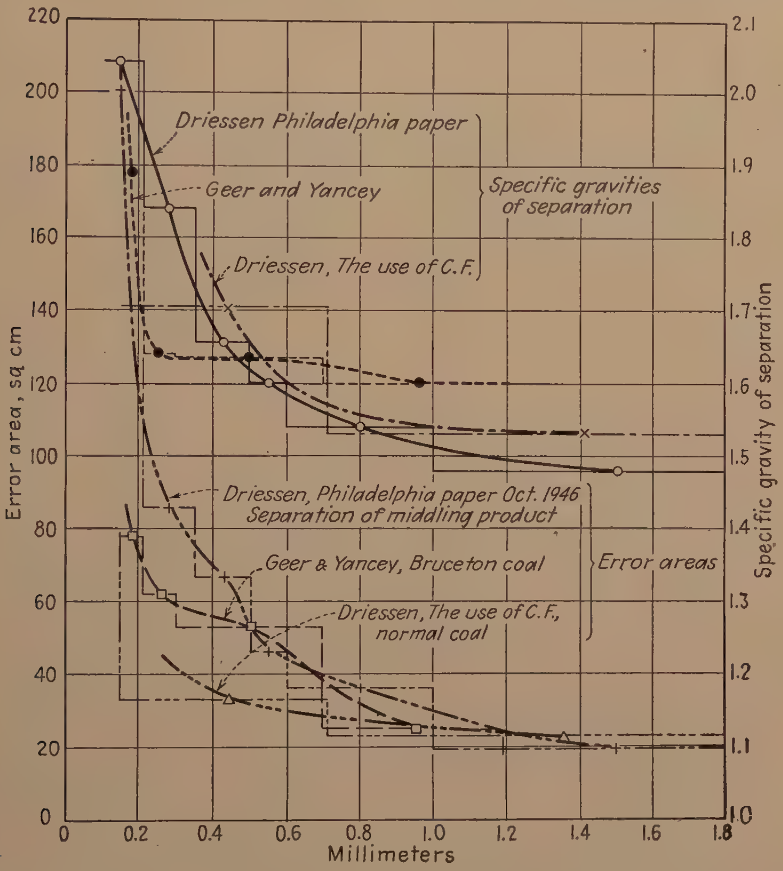

FIG 8—CHARACTERISTICS OF CYCLONE WASHERS FOR COAL BELOW ONE MILLIMETER (16-MESH).

Although I have made some remarks of minor importance referring to the methods of comparing the results of the cyclone washer, Geer and Yancey's tests agree qualitatively with the results of the Dutch State Mines, and a valuable contribution has been delivered in respect to the factors that control the operating characteristics of the cyclone washer. The very important question, "How far down in size can coal be washed satisfactorily in a cyclone washer?" is illustrated in Fig 8, which also shows that Geer and Yancey's results show the same tendency as the State Mine data. It seems that a washing effect is present down to 0.1 mm (140-mesh) although the separation is no more exact (error area, 200 sq cm), and the specific gravity of separation may increase to 2.0. However, very often the ash content is not affected too much by a high separation of the finer fractions, a fact that is clearly illustrated by the report of Fraser and Crentz on low-gravity separation of American coals.

W. J. PARTON[*]—The development of the cyclone washer described is of considerable interest since it affords another system for handling the fine sizes that generally present the most difficulty in coal cleaning. With the increasing interest and development of heavy-media cleaning systems in this country, the cyclone should be a welcome tool to supplement the cleaning equipment for the larger sizes, since it extends greatly the range of sizes amenable to treatment by existing heavy-media processes. In fact, the installation of a cyclone washer for fine sizes would offer the most advantage at a plant where a heavy-medium process is employed, so that the same preparation, reclamation and cleaning facilities could be used for the medium of both systems. In this way, the main disadvantages of the cyclone washer could be eliminated.

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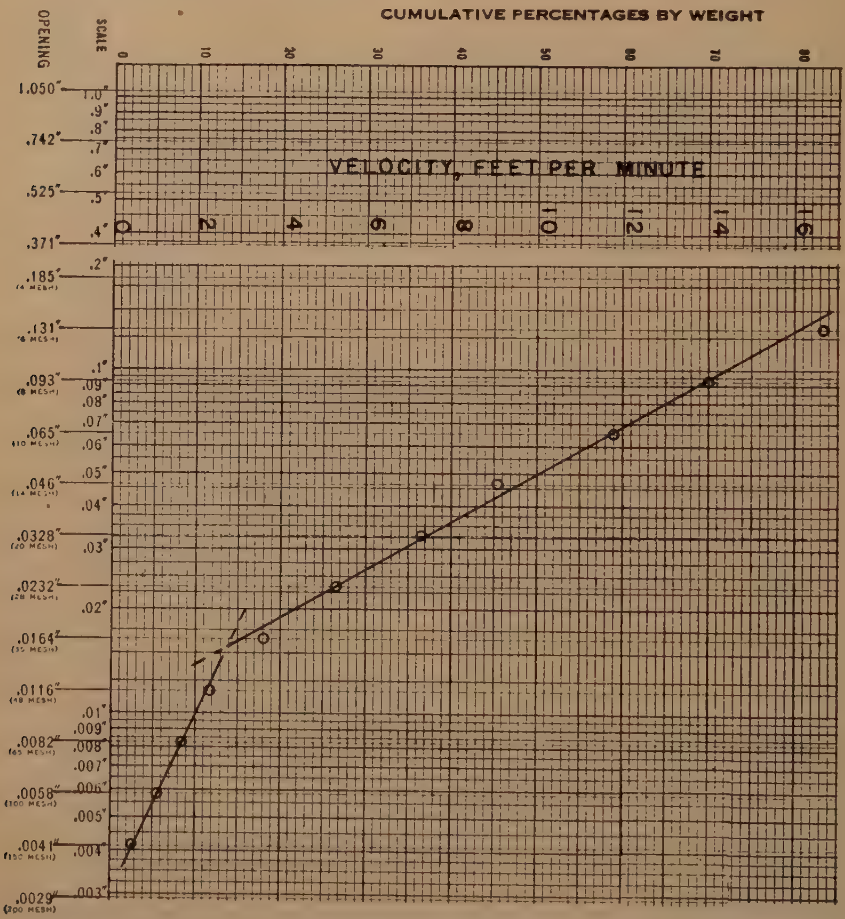

Fig 9—Tests by Colorado Fuel and Iron Corporation on place of change in behavior of small particles.

Tyler Standard Screen scale. Plotted points are for average values based on a minimum of ten trials.

Test data indicate that the cyclone washer is capable of cleaning particles efficiently to a lower size limit of approximately 48-mesh. If the washer is to be used for cleaning to this small size, one of the problems will be to reclaim the medium from the washed coal and the refuse products. Screening on a commercial scale is not very successful below 28-mesh.

Cleaning of the medium to prevent undue contamination is also one of the major problems to be considered in the selection of the medium for the process. For that reason, it appears that magnetite would be the most suitable medium as its magnetic property makes possible an efficient means of cleaning. If flotation must be employed to clean the suspension, the process becomes more complicated and expensive.

Although the tests described in this paper were conducted on minus 14-mesh particles, the system should be capable of cleaning raw-coal feeds containing much larger top-size material. This would be a considerable advantage, since screening the raw feed into fractions for separate treatment would be more efficient at a larger size cut.

W. M. Bertholf*—The authors are to be congratulated on an excellent paper dealing with a subject that is of more than academic interest; viz., the cleaning of fine coal. The possibility of making an effective separation of fine coal and refuse in a simple mechanism deserves further study.

One point appears not to have been studied directly in the preliminary tests reported, and it may prove rather difficult to analyze; but there must be some relationship between the acceleration developed in the cyclone and the nature of the separation effected. Data given in the paper indicate that with higher tonnage rates the efficiency and separating gravity both increase. Obviously there is no advantage to making a perfect separation (?) at a gravity so high that no reject is removed, hence we cannot assume that the increase in efficiency with greater acceleration is entirely a blessing. However, if it were possible to determine the *G*'s developed and to correlate this number with performance it might simplify the matter of translating results obtained in the small cyclone to what could be expected in larger (commercial) units.

We are not at all surprised that the cyclone's efficiency drops as the particle size goes below 35-mesh. Some experiments to determine the free-settling velocity of coal in the size range from 6 to 150-mesh showed a very sharp change in the "velocity *vs.* particle size" relationship at 35-mesh. Our conclusion was that it was unlikely that any process except froth flotation would be able to treat minus 35-mesh coal at acceptable tonnage rates and efficiencies, hence the handling of coal in such a way as to minimize the production of extremely fine sizes is usually desirable. (Fig 9.)

The problem of recovering the fine coal (and refuse) from the suspending medium after the separation has been accomplished is, in our opinion, crucial. If a nonmagnetic medium is used and froth flotation must be employed to effect the recovery of fine coal, are we not, in reality, doing the same thing twice? Would it not be just as easy to make the separation in the flotation machine? Our experience indicates that it is not difficult to float fine coal; but it is difficult to handle the froth. A satisfactory solution to the problems of breaking down and dewatering the froth is needed. The recovery of fine coal from a magnetic medium appears (in the literature, at least) to be relatively simple—at least one does not have to work with the flotation froth—and we would guess that the cyclone has more chance of succeeding if used with a magnetic suspending medium.

Ordinarily we consider coal a low-grade ore and try to keep our milling practice as simple and cheap as possible; but there are probably a number of places where such an apparatus as the cyclone, together with the required auxiliary machinery, can be used to advantage. If the cost of coal continues to increase, the *necessity* for more elaborate treatment will also increase. Studies such as this certainly are not a waste of time and effort, even though conducted on small-scale models.

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The Use of Hydraulic Cyclones as Thickeners and Washers in Modern Coal Preparation

By M. G. Driessen*

(Philadelphia Meeting, October 1946)

For a number of years the cyclone, familiar to Americans as a dust collector, has been used as a thickener of suspensions at one of the coal-preparation plants of the Netherland State Mines in Limburg. The plant treats nut-size coal at 120 tons an hour in a heavy-medium washer. The diluted medium from the rinsing screens is thickened in cyclones. More recently the cyclone has been adapted to act as a coal cleaner.

The principle of the cyclone thickener is the same as that of the cyclones used in industry for catching dust or fine particles suspended in the air or in gases. Generally the latter are more efficient in this respect—the smallest particles that can be caught with reasonable efficiency are considerably smaller than with the cyclone thickeners. The reason is obvious; the ratio of specific gravities between the dust particles in a gas and the gas itself is about 800 times larger than the ratio between, for instance, the specific gravities of shale particles and water. The ratio of the velocities of fall of shale particles ($\gamma = 2.5$)† in air and in water is as follows:

In the range of Stokes: $w_{air} = 93w_{water}$

In the range of Allen: $w_{air} = 49w_{water}$

In the range of Newton-Rittinger: $w_{air} = 36.6w_{water}$

The same ratios apply for the velocities in the centrifugal field, from which it follows that it is especially difficult to catch the very fine suspension particles in a liquid. The efficiency of a cyclone thickener depends mainly on the circumferential velocity of the liquid at its innermost radius, because here the centrifugal forces are the highest. However, these velocities cannot easily be calculated, as the flow is three-dimensional, and thus the axial components cannot be neglected (p. 252, condition 3). We believe that at first a spiral tangential motion along the cyclone wall is created, which causes the inner vortex ascending from the apex of the cyclone (Fig 1). At the same time the inner vortex is strengthened by the individual vortices in horizontal planes, and thus a part of the total flow can find another and perhaps quicker way to the outlet. In any case, the exact motion, not only of the fluid but especially of the suspended particles, is so difficult that we have to refer to the experiments to find the exact behavior of this kind of apparatus.

Method of Judging Performance of Coal Washer

Geologists, rather than mechanical engineers, would be able to describe the separating or stratifying effect in nature, but even a bather on the shore lying on the soft sand near the sea wonders why all the small sand grains have approximately the same size. A child playing on the border of the river could easily

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† In this paper γ represents specific gravity.

collect a hundred pebbles of practically the same diameter. When a mine shaft is sunk the overburden is often divided into horizontal strata of sands of different colors and particle sizes. In our country (The Netherlands, Limburg) and in China thick layers of so-called loess cover the earth; its main characteristic is that the sizes of the particles, although considerably smaller than those of sand, are limited within very small boundaries.[1] The theory generally adopted for loess is that it is an air-borne material, which has been deposited on quiet places.

In nature, the separating or stratifying effect seems to be more influential than a mixing effect, and this can be understood as soon as a similar law is applied to particles of different physical properties. Thus, for instance, if silica particles of different sizes are swept away by an upward air current, the coarser sizes will drop immediately whereas the smaller sizes can be carried a considerable distance, with the wind, in accordance with Stokes' law (Eq 20).

In Bureau of Mines *Bulletin* 300, pages 76 and 119, use has been made of the "Hancock efficiency charts," in which the recoveries or efficiencies of the product (coal) are plotted as abscissas and the different specific-gravity fractions of the raw coal as ordinates. In order to compare the separation results of different tests, we prefer to plot the recoveries in function of the value by which the respective physical property can be measured. Thus the abscissas in the case of a specific-gravity separator (wash box) are the specific gravities themselves; not the specific-gravity fractions as percentages of the feed. By way of comparison, Fig 2, with the tables, shows at the right and left the recoveries for two different coals plotted in a similar way, as on the Hancock efficiency charts but turned 90°, whereas the middle figure shows the recoveries plotted in the new way. The latter curve is called the relative-error curve, or Tromp curve if the upper part, starting from 50 pct recovery, is inverted 180°, and the area underlying these two sections of the error curve is called the error area. The error area measures the faults compared with ideal washing, when the separation can be illustrated by a vertical line, indicating at the left 100 pct recovery of the fractions lighter than the specific gravity of separation and at the right 0 pct recovery for the fractions heavier than the specific gravity of separation.

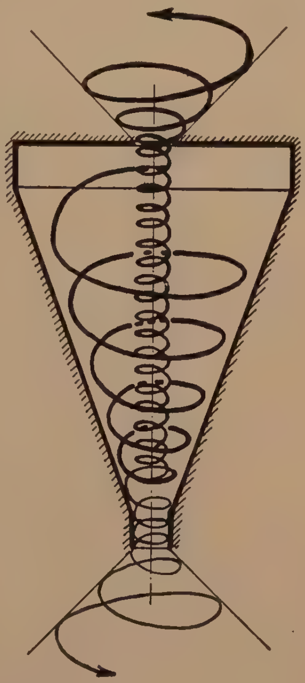

FIG 1—OUTER AND INNER VORTEX IN CYCLONE.

EFFICIENCY OR RECOVERY CURVES OF CYCLONES

A cyclone used in washery practice can be looked at from different viewpoints. The particle size of the suspension or the specific gravity of the particle

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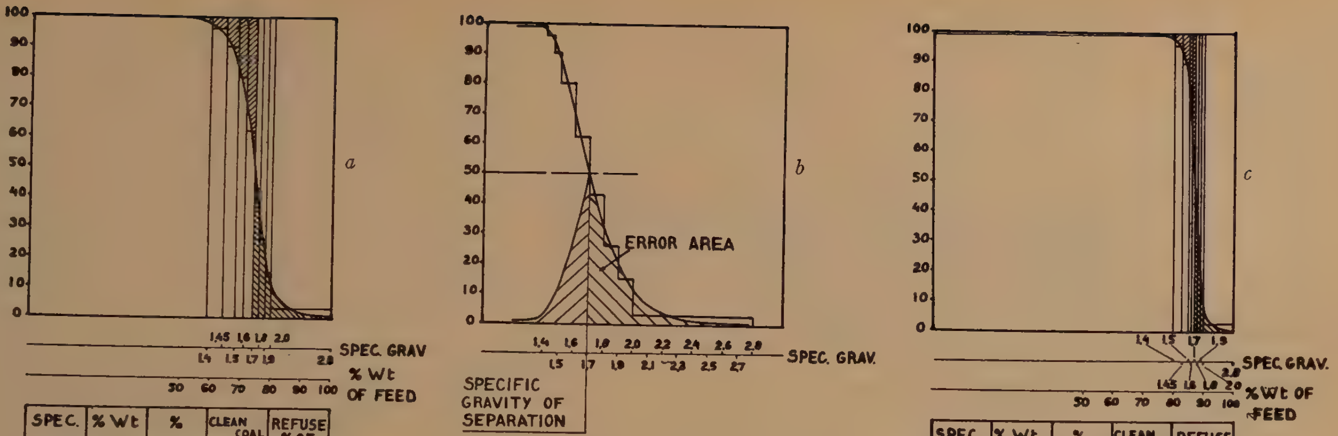
[1] References are at the end of the paper.



SPEC. GRAVITY	% Wt OF FEED	% RECOVERY	CLEAN COAL % OF FEED	REFUSE % OF FEED
<1,4	59	99	58,41	0,59
1,4–1,45	5	96	4,80	0,20
1,45–1,5	4	90	3,60	0,40
1,5–1,6	3	80	2,40	0,60
1,6–1,7	3	62	1,86	1,14
1,7–1,8	2	43	0,86	1,14
1,8–1,9	2	26	0,52	1,48
1,9–2,0	2	15	0,30	1,70
>2,0	20	3	0,60	19,40
	100		73,35	26,65

REFUSE: FLOAT ON SPEC. GRAVITY OF SEPARATION $(1{,}7) = \frac{0{,}59+0{,}2+0{,}4+0{,}6+1{,}14}{26{,}65} \times 100 = 11\%$

CLEAN COAL: SINK ON SPEC. GRAVITY OF SEPARATION $(1{,}7) = \frac{0{,}86+0{,}52+0{,}3+0{,}6}{7{,}35} \times 100 = 3{,}1\%$

SPEC. GRAVITY	% Wt OF FEED	% RECOVERY	CLEAN COAL % OF FEED	REFUSE % OF FEED
<1,4	80	99	79,20	0,80
1,4–1,45	3	96	2,88	0,12
1,45–1,5	2	90	1,80	0,20
1,5–1,6	1	80	0,80	0,20
1,6–1,7	1	62	0,62	0,38
1,7–1,8	1	43	0,43	0,57
1,8–1,9	1	26	0,26	0,74
1,9–2,0	1	15	0,15	0,85
>2,0	10	3	0,30	9,70
	100		86,44	13,56

REFUSE: FLOAT ON SPEC. GRAVITY OF SEPARATION $(1{,}7) = \frac{0{,}8+0{,}12+0{,}2+0{,}2+0{,}38}{13{,}56} \times 100 = 12{,}5\%$

CLEAN COAL: SINK ON SPEC. GRAVITY (OF SEPARATION $(1{,}7) = \frac{0{,}43+0{,}26+0{,}15+0{,}3}{86{,}44} \times 100 = 1{,}3\%$

FIG 2—GRAPHS FOR ILLUSTRATING EFFICIENCY OR RECOVERY OF GRAVITY-SEPARATING VESSELS (WASH BOXES).
a and *c*, absolute error curves, similar to Hancock chart.
b. Relative error curve and Tromp curve (upper part inverted 180°).

introduced into the cyclone, or both, may be the physical property for which a separation is required. In the first case we have a cyclone thickener, where a suspension in water or other liquid is introduced into the cyclone, and where a separation takes place between the coarser and the finer suspension particles. In the cyclone washer we are interested in the separation of particles according to their specific gravity, but as we introduce these particles together with a suspension, a separation of the suspension particles according to size takes place at the same time. It may be described also in this way: Superposed on the thickening effect of the suspension particles is the washing effect of the coal and shale particles, which are introduced together with the suspension.

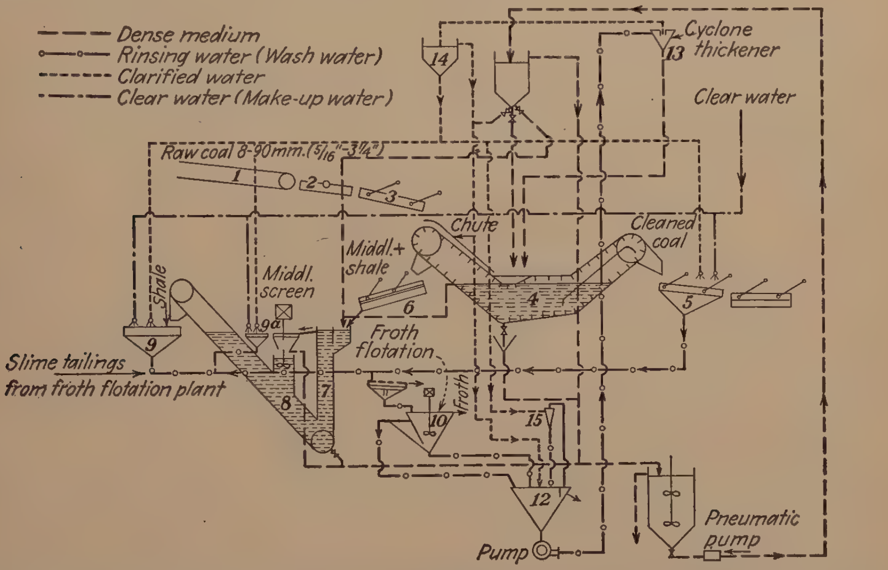


FIG 3—FLOWSHEET WITH CYCLONES IN HEAVY-MEDIUM WASHERY, NETHERLAND STATE MINE MAURITS.

Slime tailings 120 tons per hour, coal 8 to 90 mm.

1.	Belt conveyor.
2, 3.	Vibrating screens to remove undersize.
4.	Heavy-medium wash box.
5, 9, 9*a*.	Rinsing screens for coal, shale and middlings.
6.	Three-size classifying screen.
7.	Upward-current box.
8.	Elevator for shale.
10.	Frother.
11.	Vibrating screen to remove coarse particles.
12.	Pump tank.
13.	Main cyclone thickeners.
14.	Rinsing-water tank.
15.	Auxiliary cyclone thickeners for effluent.

EXAMPLE OF CYCLONE THICKENERS

Fig 3 shows a flowsheet of a suspension washery working at the rate of 120-ton per hour on slime tailings from the froth-flotation plant at the Netherland State mine, Maurits, in which a cyclone thickener is used. Raw 8 to 90-mm coal (5⁄16 to 3¼ in.) is fed to the plant by a belt conveyor 1 and screened to remove the undersize by vibrating screens 2 and 3.

From there the coal is passed to heavy-medium wash box 4, provided with a scraper to remove the clean coal to the rinsing screen 5 and the shale plus middlings to a vibrating screen 6. From the latter the screened sizes are fed to an upward-current box 7, consisting of three compartments in which the final shale is separated from the middlings and allowed to sink to elevator 8, from which it is forwarded to rinsing screen 9. The middlings overflow with the upward current to the middlings screen 9*a*.

Circulation and Recovery of Rinsing Water and Suspension

The rinsing water, enriched with the heavy suspension from the coal, middlings and shale, flows to a vibrating screen 11, where the coarser coal and shale particles are screened. From there the water flows to the frother 10, where the fine coal particles are eliminated. The regenerated tailing water flows to the pump tank 12. From there the suspension is pumped to the cyclone thickeners 13, and is thickened to the required specific gravity of the wash box. The clarified water of the cyclones 13 flows to the rinsing-water tank 14, from which it is distributed to the screens 5, 9 and 9*a*. Part of the rinsing water flows to a smaller and more efficient cyclone 15, the overflow of which is better clarified than that of the main cyclones 13, so that it may flow to waste as an effluent of the system. The thickened material of the small cyclone 15 is returned to the circulating flow system. Clear water is admitted to screens 5, 9, and 9*a* for final rinsing of the washed products.

The behavior of the primary cyclones 13 has been carefully investigated during operation of the plant, and the results are compiled in Table 1. Fig 4 shows the recovery of the solid particles in the thickened suspension as a function of their respective diameters. This curve may be called an error curve, as the ideal separation would be a vertical line showing that the recovery left of this line is zero, indicating that no particles under a certain size can be found in the thickened suspension, whereas the recovery on the right of the vertical would show that all particles larger than the separating size would be caught in the thickened suspension. We will call the particle size of which 50 pct goes to the clarified water and 50 pct to the thickened suspension the separating size of the apparatus. In our cyclone thickeners, the separation size is approximately 35 μ. For other cyclones,

TABLE 1—*Size Distribution and Efficiencies (Recoveries) of Size Fractions for Cyclone Thickener 13 of Heavy-medium Washery*

DATA	CYCLONE THICKENER
Diameter of cyclone, mm	350
Water pressure, atm	2.5
Quantity feed, cu meters per hr	50.5
Quantity clarified water, cu meters per hr	46.25
Quantity thickened suspension, cu meters per hr	4.25
Solid concentration feed, grams per liter	139
Solid concentration clarified water, grams per liter	60
Solid concentration thickened suspension, grams per liter	986
Specific gravity of thickened suspension, kg per liter	1.555
Particle size of feed, microns pct:	
+500	2.5
350–500	2.9
210–350	7.0
105–210	16.9
50–105	18.8
− 50	51.9
Particle size of clarified water, microns, pct:	
+500	0.2
350–500	0.4
210–350	0.8
105–210	1.7
50–105	5.4
− 50	91.5
Particle size of thickened suspension, microns, pct:	
+500	4.1
350–500	4.5
210–350	11.1
105–210	27.0
50–105	27.7
− 50	25.6
Efficiency (recovery) of individual size fractions in thickened suspension, pct:	
+500	96.7
350–500	95.6
210–350	95.3
105–210	96.2
50–105	88.8
− 50	29.9
Average recovery of solids in thickened suspension, pct	60.2

it depends on the shape and diameter of the cyclone, the inlet and outlet opening, the admission pressure, the solid contents of the feed, and other conditions. The size-distribution curves for the feed, the thickened suspension and the clarified water are shown on Fig 5.

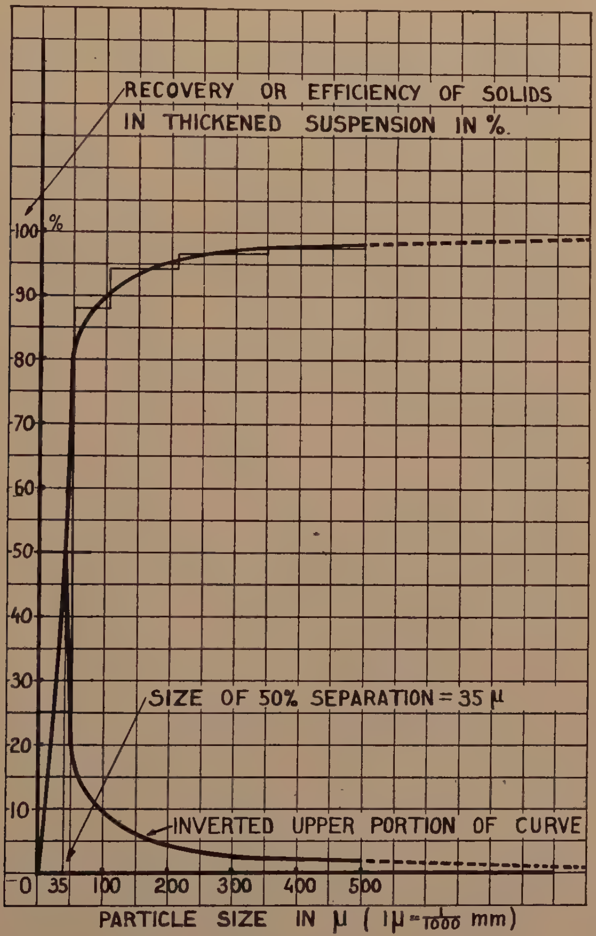


Fig 4—Recovery or error of cyclone thickeners used in slime-tailings suspension washery.

Example of Cyclone Washer

The method and apparatus necessary to clean fine coal in a cyclone washer were described in an earlier paper,[2] and the corresponding flowsheet is reproduced in this paper in Fig 6.

Raw fine coal is admitted from a coal hopper by means of a belt conveyor into container B, filled with the suspension of the desired specific gravity. An air lift pumps the mixture to a vessel V, placed about 13 m (43 ft) above the cyclone washer. This height is sufficient to furnish the necessary inlet pressure for cyclone C_W. The air lift is installed to take care of local conditions at the laboratory. An actual plant would provide for a bucket elevator to bring the fine coal directly to vessel *V*.

The separated products, coal and shale, leaving the cyclone are discharged on two vibrating screens *S*. Under each screen two fluid receivers *F* are placed; the first takes the undiluted suspension and returns it to container *B* and the second takes the rinsing water, enriched with the suspension washed off over the second part of the screen. The rinsing water then passes to a rotating screen *R* to remove the coarser coal and shale particles and arrives in a tank *T*, from which the pump P_p, delivers the rinsing water to the cyclone thickener C_t.

The clarified water from the cyclone thickener C_t is used as rinsing water on the screen *S*, but additional fresh water is sprayed onto the screen below the first rinsing.

The amount of fresh water must leave the installation at another point, and thus part of the clarified water is admitted to secondary cyclone C_s, where an additional clarification takes place. The thickened suspension is returned to the system, and the twice-clarified water flows to waste.

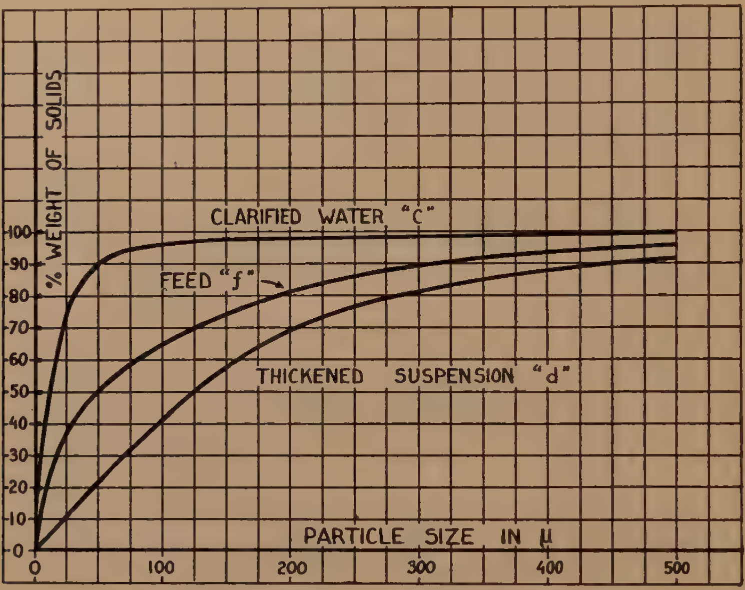


FIG 5—SIZE DISTRIBUTION OF CYCLONE THICKENERS.

Fig 6 also shows the supply S_y of heavy material for the suspension; a mixer *M* to suspend the heavy material in water, and a pressure vessel *P* to raise the suspension to container *B*.

The results of the various tests are plotted in a way similar to that described for the cyclone thickener. For the cyclone washer, the recoveries or efficiencies of the specific-gravity fractions are plotted against the corresponding specific gravities, and the upper part of this curve is inverted 180° at the 50 pct recovery axis. Characteristic of the washing effect of the cyclone is the error area, which can be expressed in square centimeters; if on the abscissa 0.1 pct increase of specific gravity corresponds to 1 cm, and on the ordinates 1 pct recovery corresponds to 0.5 cm.

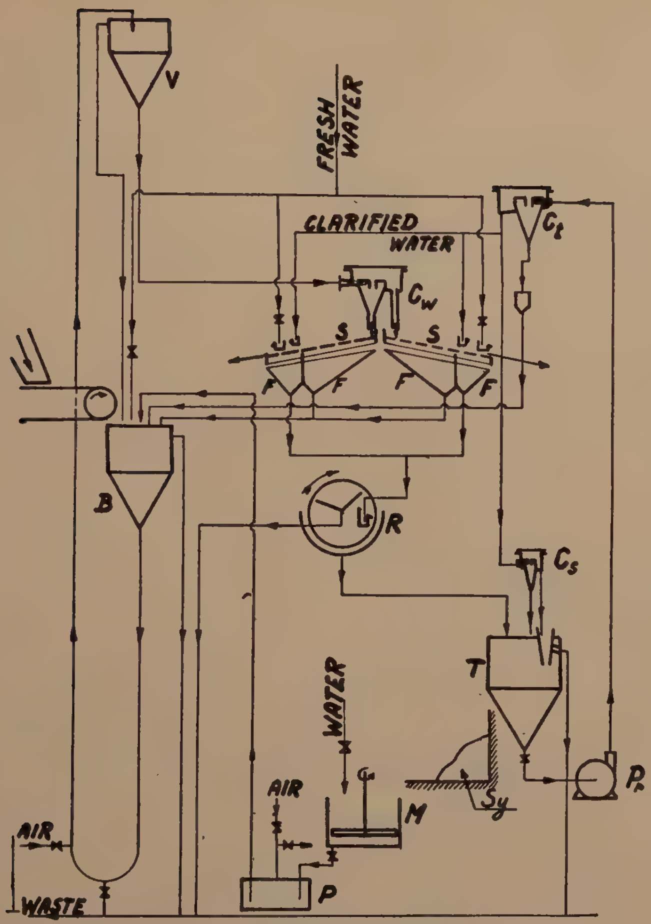


FIG 6—PILOT CYCLONE WASHERY FOR FINE COAL.

B	Suspension container.
V	Suspension vessel.
C_w	Cyclone washer.
S	Vibrating screens.
F	Fluid receivers.
T	Rinsing-water tank.
R	Rotating screen.
C_s	Secondary cyclone.
P_p	Pump.
S_y	Heavy-medium supply.
M	Suspension mixer.
P	Pressure vessel.
C_t	Cyclone thickener.

In order to study the washing effect of the cyclone for a very bad coal, we have treated a fine-coal middling product of 44.5 pct ash in the cyclone washer and obtained results that are compiled in Table 2. Table 3 shows the analysis of the feed for the various size fractions. The data indicate that the feed, which contains only 26.4 pct of material lighter than 1.45 sp gr, can be separated into a clean coal of 6 pct ash and a refuse of 56.1 pct ash, whereas the float at 1.45 in the refuse is only 0.8 pct. The feed was screened on ½ mm, but still contained 15.14 pct under this size.

The curve of Fig 7 shows that a separation very near to theoretical is effected down to ½-mm, and that the error area, as well as the specific gravity of separation, increases rapidly under this size. Where a very exact separation is necessary, for instance, when extremely low-ash coal is desired, the washed products can be screened at ½ mm. This, of course, does not apply to the example just given, but only to cases where the feed is already low in ash and contains a sufficiently high percentage of low-gravity and low-ash coal. In order to find the relationship between the error area, expressed in square centimeters, and the actual losses resulting in inexact separation for a certain feed, Fig 11 of ref. 2 is reproduced here as Fig 8. The losses are the difference in yield between theoretical separation at a certain ash content and the actual yield

TABLE 2—*Effect of Grain Size on Error Area and Specific Gravity of Separation in Cyclone Washer Treating Middling Product* 0 *to* 10 *Millimeters of* 44.5 *Per Cent Ash*

Grain Size, Mm	Error Area, Sq Cm	Specific Gravity of Separation	Feed, Pct		Coal, Pct		Refuse, Pct	
			Weight	Ash	Weight	Ash	Weight	Ash
+8	9.8	1.48	5.04	36.3	4.9	5.3	5.1	49.0
4 –8	13.7	1.46	27.42	40.8	16.4	5.6	32.1	49.8
2 –4	15.6	1.46	20.41	40.0	18.6	5.6	21.2	55.1
1 –2	19.2	1.48	23.21	46.6	20.7	6.5	24.2	63.7
0.6 –1	36.3	1.54	6.58	50.5	6.6	6.7	6.5	72.2
0.5 –0.6	46.1	1.60	2.19	39.9	3.1	6.5	1.8	72.2
0.35 –0.5	66.3	1.71	4.65	44.4	7.1	8.6	3.6	72.7
0.21 –0.35	85.8	1.84	4.33	33.5	8.6	13.4	2.5	76.1
0.105–0.21	200.	2.08	6.16	48.5	14.0	33.2	2.8	81.2
			100.00		100.0		100.0	
Weights of products + 0.5 mm, kg per hour			6,146		1,625		4,521	
Ash content of products + 0.5 mm, pct			44.5		6.0		56.1	
Ash content of products 0–10 mm, pct			45.8		10.6		58.5	
Floats on 1.45 sp gr products + 0.5 mm, pct			26.4				0.8	

TABLE 3—*Analyses of Feed to Cyclone Washer*[a]

Specific Gravity Fractions	Size Fractions								
	+8 Mm	4 × 8 Mm	2 × 4 Mm	1 × 2 Mm	0.6 × 1 Mm	0.5 × 0.6 Mm	0.35 × 0.5 Mm	0.21 × 0.35 Mm	0.105 × 0.21 Mm
–1.3	5.07	3.66	7.17	8.00	9.55	13.26	23.01	22.79	11.19
1.3 × 1.35	12.60	8.56	11.55	8.12	9.22	8.10	11.33	14.92	19.93
1.35 × 1.4	4.23	4.44	4.27	5.20	5.96	7.68	6.03	10.44	7.38
1.4 × 1.45	1.83	2.44	3.11	2.26	2.86	1.09	1.77	1.96	1.67
1.45 × 1.5	2.26	4.92	4.37	2.48	2.28	1.32	1.33	2.28	1.37
1.5 × 1.55	4.50	7.46	4.90	2.63	1.50	1.69	1.59	1.98	1.58
1.55 × 1.6	8.70	7.38	33.38	3.23	1.50	1.27	1.97	.78	.70
1.6 × 1.7	17.60	12.26		5.88	2.72	2.47	2.44	2.28	1.58
1.7 × 1.8	9.68	10.40		15.40	2.40	2.02	1.85	3.47	1.70
1.8 × 1.9	7.35	9.12			1.99	3.40	2.50	2.03	1.36
1.9 × 2.0	5.93	6.72			2.31	6.24	5.23	1.90	2.85
+2.0	[illegible]0.25	[illegible].64	31.25	46.80	57.72	51.47	41.03	35.20	48.71
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Percentage of Feed	5.04	27.42	20.41	23.21	6.58	2.19	4.65	4.33	6.16

[a] For results see Table 2.

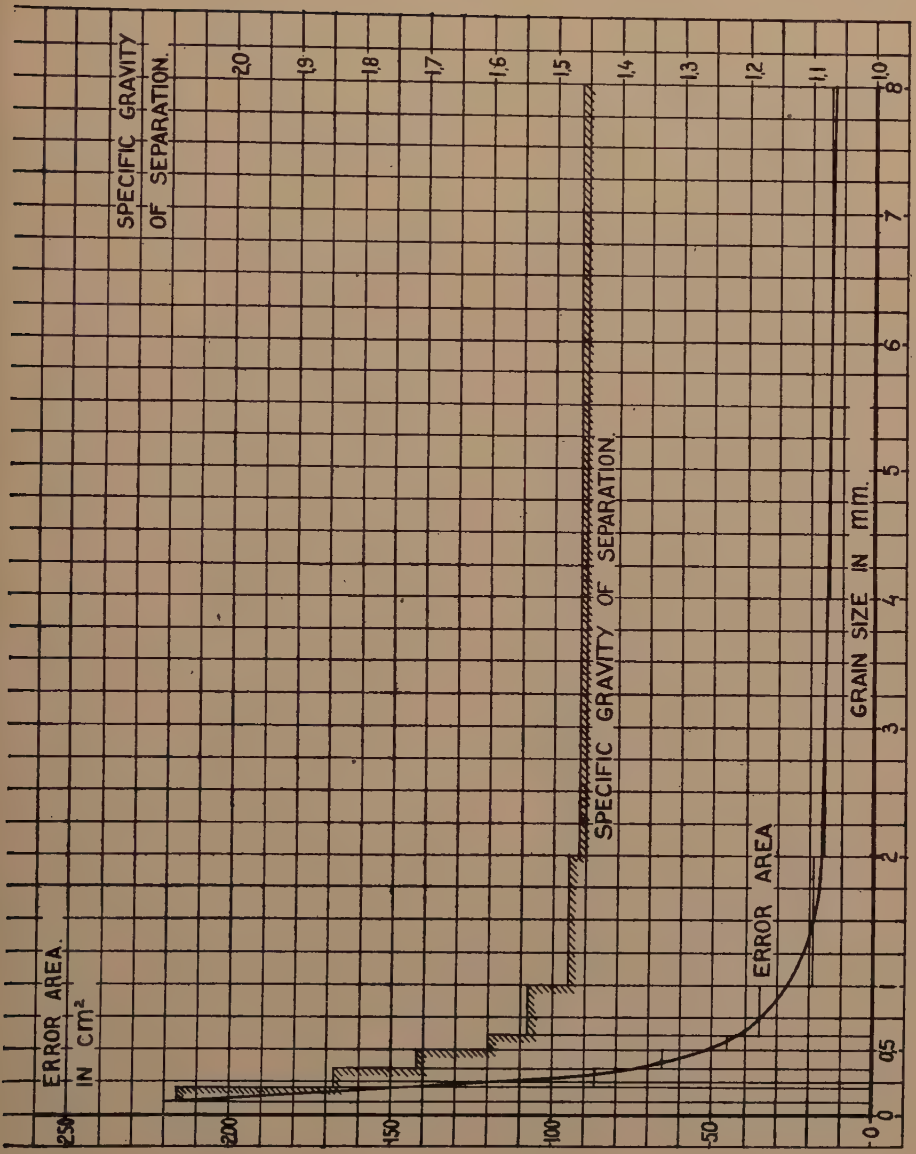
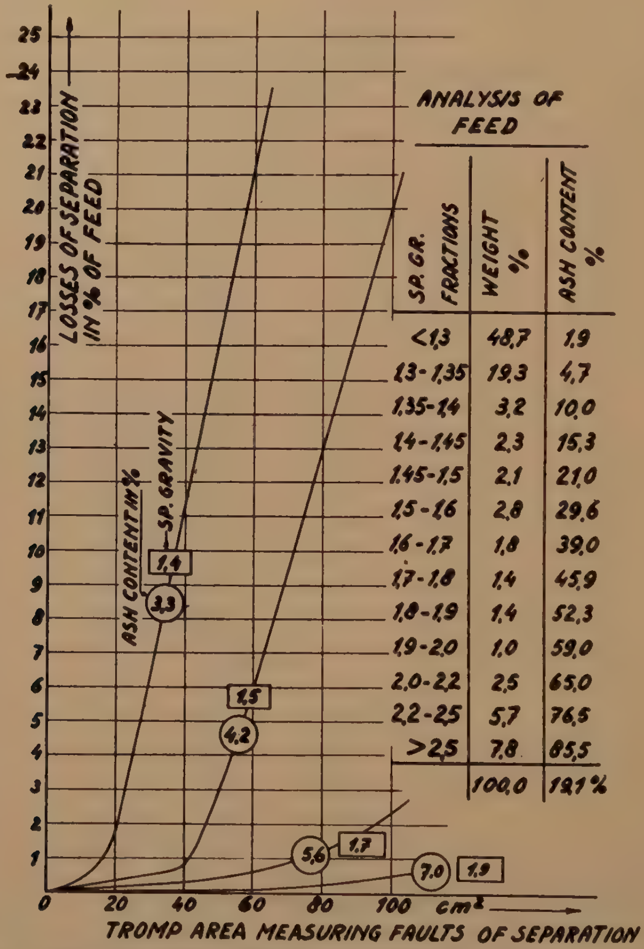


Fig 7—Error area and specific gravity of separation in function of grain size. Feed of middlings 0.5 to 10 mm with 26.4 pct lighter than 1.45 sp gr and 44.5 pct ash.

at the same ash content. This corresponds with the method to compute washery results, as described by Yancey and Fraser,[3] where the actual washing is compared with float-and-sink tests (theoretical separation) and plotted against the cumulative ash percentages. The difference between the "float-and-sink" curve and "washing curve" at a certain ash percentage is the "loss of separation," just referred to, and may also be represented as an efficiency, as in Fig 37 of the publication cited.[3] This graph, as well as our Fig 8, shows a very quick increase of the losses for separation at a low specific gravity or at a low ash content, and thus for these cases an efficient cleaning apparatus such as the cyclone washer is necessary.



SP. GR. FRACTIONS	WEIGHT %	ASH CONTENT %
<1,3	48,7	1,9
1,3-1,35	19,3	4,7
1,35-1,4	3,2	10,0
1,4-1,45	2,3	15,3
1,45-1,5	2,1	21,0
1,5-1,6	2,8	29,6
1,6-1,7	1,8	39,0
1,7-1,8	1,4	45,9
1,8-1,9	1,4	52,3
1,9-2,0	1,0	59,0
2,0-2,2	2,5	65,0
2,2-2,5	5,7	76,5
>2,5	7,8	85,5
	100,0	19,1 %

FIG 8—LOSSES CAUSED BY INEXACT SEPARATION IN FUNCTION OF ERROR AREA AT DIFFERENT SPECIFIC GRAVITIES OF SEPARATION FOR FEED.

MECHANICS OF CYCLONE

Although it is not yet possible to give an exact theory about the cyclone thickener or cyclone washer, some problems directly connected with the working of a cyclone have been solved and will be explained in the following:

IDEAL TWO-DIMENSIONAL FLOW OF NONVISCOUS FLUID

The motion of the liquid in a cyclone can be described comparatively easily if an ideal nonviscous fluid is assumed.

In this case the motion can be thought of as a superposition of a vortex and a sink, each of which is a potential motion, so that the potential functions can be added together.

Fig 9*a* shows the streamlines and the velocities of a vortex with circulation K and increasing tangential velocities v toward the center but without vorticity, in areas that exclude the center of rotation.

The potential function, written in cylindrical polar coordination r and φ is:

$$\phi_1 = \frac{K}{2\pi}\varphi \qquad [1]$$

and the tangential velocities can be found as follows:

$$v = \frac{\partial\phi_1}{r\partial\varphi} = \frac{K}{2\pi r} \qquad [2]$$

Fig 9*b* shows the streamlines and the velocities of a sink with a constant capacity $-e$ and increasing radial velocities toward the center. The potential function is:

$$\phi_2 = -\frac{e}{2\pi}\ln r \qquad [3]$$

and the radial velocities can be found as follows:

$$u = \frac{\partial\phi_2}{\partial r} = -\frac{e}{2\pi r} \qquad [4]$$

Fig 9*c* shows the superposition of Figs 9*a* and 9*b*. The potential function is:

$$\phi = \phi_1 + \phi_2 = \frac{1}{2\pi}(K\varphi - e\ln r) \qquad [5]$$

The velocities c in the direction of the streamlines are:

$$c = \sqrt{u^2 + v^2} = \frac{1}{2\pi r}\sqrt{e^2 + K^2} \qquad [6]$$

and the direction of the streamlines with reference to radius R is given by:

$$\tan\alpha = \frac{v}{u} = -\frac{K}{e} = \text{constant} \qquad [7]$$

from which it follows that the streamlines are logarithmical spirals.

Fig 9*d* shows how a pattern actually obtained on a cover of a 350-mm cyclone, used as a cyclone thickener during several months, compares with the theoretical spirals.

It will be appreciated that the ideal nonviscous potential flow is impossible in practice because the velocities would grow to infinity when approaching the center or Z-axis. However, we will calculate the maximum tangential velocity, assuming ideal conditions for a hydraulic cyclone of 35-cm diameter, a pressure of 2 atm, and a capacity of about 36 cu m per hr, the diameter of the inlet and outlet openings being 5 cm.

The inlet velocity:

$$v_1 = \frac{36.}{\frac{\pi}{4}0.05^2 3600} = 5.1 \text{ m per sec}$$

The minimum radius r_2 at which a flow of liquid can be observed is somewhat smaller than the radius of the outlet opening, which is 2.5 cm. From Eq 2 we find that:

$$vr = \text{constant} = v_1 r_1 = v_2 r_2$$

in our case,

$$v_1 = 5.1 \text{ m per sec}$$
$$r_1 = 0.15 \text{ m}$$

so that $r_2 < 0.025$ m

$$v_2 > 30.6 \text{ m per sec}$$

and the ratio between the centrifugal force of a particle turning with velocity v_2 at a radius r_2 and gravity is

$$\frac{\text{Centrifugal force}}{\text{Gravity}} = \frac{v_2^2}{r_2 g} = 3800$$

Flow of Viscous Fluid

The general equations for a viscous flow are those given by Navier-Stokes and for our case the most apt for calculation are the cylindrical polar coordinates, written as follows:

$$\frac{\partial u}{\partial t} + u\frac{\partial u}{\partial r} + \frac{v}{r}\frac{\partial u}{\partial \varphi} + w\frac{\partial u}{\partial z} - \frac{v^2}{r} = -\frac{1}{\rho}\frac{\partial p}{\partial r} + \nu\left(\nabla^2 u - \frac{u}{r^2} - \frac{2}{r^2}\frac{\partial v}{\partial \varphi}\right)$$

$$\frac{\partial v}{\partial t} + u\frac{\partial v}{\partial r} + \frac{v\partial v}{r\partial \varphi} + w\frac{\partial v}{\partial z} + \frac{uv}{r} = -\frac{1}{\rho}\frac{\partial p}{r\partial \varphi} + \nu\left(\nabla^2 v + \frac{2}{r^2}\frac{\partial u}{\partial \varphi} - \frac{v}{r^2}\right) \qquad [8]$$

$$\frac{\partial w}{\partial t} + u\frac{\partial w}{\partial r} + \frac{v\partial w}{r\partial \varphi} + w\frac{\partial w}{\partial z} = -\frac{1}{\rho}\frac{\partial p}{\partial z} + \nu\nabla^2 w$$

and the operator

$$\nabla^2 = \frac{\partial^2}{\partial r^2} + \frac{1}{r}\frac{\partial}{\partial r} + \frac{1}{r^2}\frac{\partial^2}{\partial \varphi^2} + \frac{\partial^2}{\partial z^2}$$

In this set of equations,

u = radial velocity,
v = tangential velocity,
w = velocity in z-axis

For our case of a polar symmetrical steady flow these equations can be considerably simplified when the following conditions are observed:

1. The flow is steady, which means that velocities and pressures are the same at a time t and at a time $t + dt$; thus

$$\frac{\partial u}{\partial t} = 0, \quad \frac{\partial v}{\partial t} = 0, \quad \frac{\partial w}{\partial t} = 0$$

2. The flow is polar symmetrical, which means that velocities and pressures remain unchanged on coaxial circles, thus:

$$\frac{\partial u}{r\partial \varphi} = 0 \qquad \frac{\partial w}{r\partial \varphi} = 0$$

$$\frac{\partial v}{r\partial \varphi} = 0 \qquad \frac{\partial p}{r\partial \varphi} = 0$$

3. Although the radial and tangential velocities u and v may change for variable z values, we assume that the flow is two dimensional in superposed horizontal planes, so that no velocity components in the z directions are present.

Thus: $w = 0$ in the whole field and also the derivatives of w disappear.

4. The fluid being assumed incompressible we have, in addition to the Navier-Stokes equations, the condition of continuity

$$\frac{\partial(ur)}{\partial r} = 0 \qquad [9]$$

This equation is the same as Eq 4 and shows that the amount of fluid flowing through coaxial circles (or cylinders) is a constant and equals $-e$.

Substituting conditions 1 to 3 in the general equations 8, we obtain the following equations for our special case:

$$\frac{u\partial u}{\partial r} - \frac{v^2}{r} = -\frac{1}{\rho}\frac{\partial p}{\partial r} + \nu\left[\frac{\partial^2 u}{\partial r^2} + \frac{\partial u}{r\partial r} + \frac{\partial^2 u}{\partial z^2} - \frac{u}{r^2}\right]$$

$$\frac{u\partial v}{\partial r} + \frac{uv}{r} = \nu\left[\frac{\partial^2 v}{\partial r^2} + \frac{\partial v}{r\partial r} + \frac{\partial^2 v}{\partial z^2} - \frac{v}{r^2}\right]$$

which can also be written as follows:

$$\frac{u\partial u}{\partial r} - \frac{v^2}{r} = -\frac{1}{\rho}\frac{\partial p}{\partial r} + \nu\left[\frac{\partial}{\partial r}\left\{\frac{1}{r}\frac{\partial(ur)}{\partial r}\right\} + \frac{\partial^2 u}{\partial z^2}\right] \qquad [10a]$$

$$u\frac{\partial(vr)}{r\partial r} = \nu\left[\frac{\partial}{\partial r}\left\{\frac{1}{r}\frac{\partial(vr)}{\partial r}\right\} + \frac{\partial^2 v}{\partial z^2}\right] \qquad [10b]$$

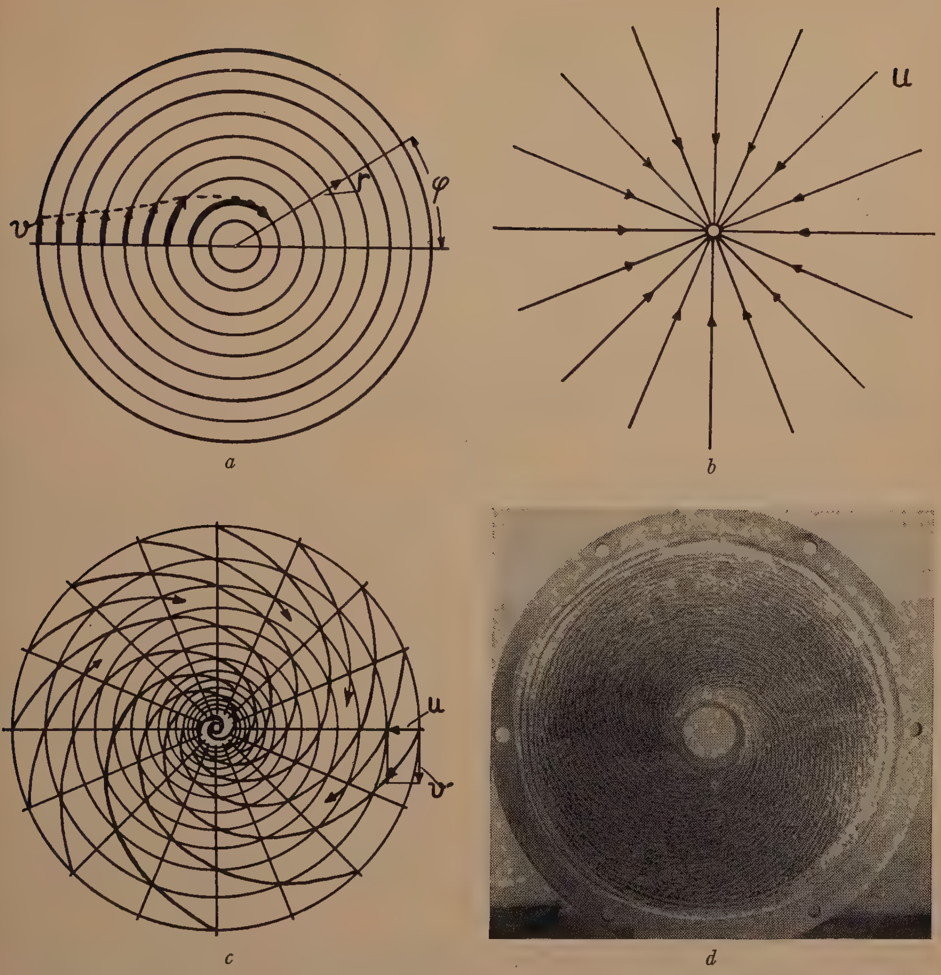


Fig 9—Superposition of vortex flow and sink to cyclonic motion of nonviscous fluid, and comparison of latter with actually obtained pattern on cover of 350-millimeter cyclone thickener.

combined with Eq 9

$$\frac{\partial(ur)}{\partial r} = 0 \qquad [10c]$$

These equations can be found in reference 5. Following the writer of that reference, we find that Eq 10*c* can be written:

$$ur = \varphi_1(z) \qquad [11]$$

an expression showing that the differential amount of fluid is a constant for a certain *z* value but may change with *z*, φ_1 being an arbitrary function of *z*.

The same equation can also be written:

$$r\frac{\partial u}{\partial r} + u = 0$$

or

$$u\frac{\partial u}{\partial r} = -\frac{u^2}{r} \qquad [12]$$

Substituting Eqs 10*c*, 11 and 12 in Eq 10*a*, we find:

$$\frac{1}{\rho}\frac{\partial p}{\partial r} = \frac{1}{r}\left((u^2 + v^2 + \nu\frac{\partial^2\varphi_1}{\partial z^2}\right) \qquad [13a]$$

We can postulate a vortex motion:

$$\frac{\partial(vr)}{\partial r} = 0$$

and

$$vr = \varphi_2(z)$$

the vortex constant being dependent only of z.

Substituting this in Eq 10*b*, we find

$$\frac{\partial^2\varphi_2}{\partial z^2} = 0 \qquad [13b]$$

so that φ_2 is a linear function of z.

The Eq 10*b* can be solved if

$$\frac{\partial^2\varphi_2}{\partial z^2} = 0$$

or

$$\frac{\partial^2 v}{\partial z^2} = 0$$

We then assume that the circulation vr is a linear function of z. If all motions in planes perpendicular to the z-axis are true vortex motions (vr = constant), we have proved that this assumption is valid.

We will see later that, apart from the motion in the close neighborhood of the core, the circular motion in viscous flows is very similar to a true vortex. Especially for the flow of normal fluids, such as water or air, the cinematic viscosity ν is so low that a practically true vortex motion is obtained.

Substituting Eq 13*b* in Eq 10*b*, and assuming that the product vr may change with varying r, so that

$$\frac{\partial(vr)}{\partial r} \neq 0$$

we find

$$u\frac{\partial(vr)}{r\partial r} = \nu\frac{\partial}{\partial r}\left\{\frac{1}{r}\frac{\partial(vr)}{\partial r}\right\} \qquad [14]$$

From Eq 10*c* we found that ur is constant in planes perpendicular to the Z-axis. For the total flow comprising all these planes, we can calculate with an average

$$ur = u_1r_1 = \frac{-Q}{2\pi B} = -A_1$$

$-Q$ being the total amount of medium passing through the cyclone and B the height in the Z direction of the entering flow at r_1, so that

$$\frac{-A_1}{r^2}\frac{\partial(vr)}{\partial r} = \nu\frac{\partial}{\partial r}\left\{\frac{1}{r}\frac{\partial(vr)}{\partial r}\right\} \qquad [15]$$

The solution of the differential equation is

$$vr = A_2r^{2-\frac{A_1}{\nu}} + A_3 \qquad [16]$$

The integration constants A_2 and A_3 can be eliminated by inserting the terminal conditions

$$vr = K \qquad \text{for } r = r_1$$
$$v = 0 \qquad \text{for } r = r_2$$

the latter condition expressing the existence of a fixed core, at which circumference the fluid is at rest.

Putting $2 - \frac{A_1}{\nu} = x$, Eq 16 will be transformed to

$$vr = K\frac{r^x - r^x_2}{r_1^x - r_2^x} \qquad [17]$$

This equation gives the tangential velocity v as a function of the radius r. There is a velocity v maximum at a certain radius r_m:

$$\frac{r_m}{r_2} = \left(\frac{1}{1-x}\right)^{\frac{1}{x}} \qquad [18]$$

v maximum can be compared with the velocity of a nonviscous flow (v nonviscous), which would occur at radius r_m:

$$\frac{v_{\text{max}}}{v_{\text{nonvis}}} = \frac{1 - \left(\frac{r_m}{r_2}\right)^x}{1 - \left(\frac{r_1}{r_2}\right)^x} \qquad [19]$$

Equations 18 and 19 are calculated on Table 4 and plotted in Fig 10 for the abscissa

$$-x = \frac{A_1}{\nu} - 2, \frac{A_1}{\nu} \text{ and } \nu$$

so that the effect of the viscosity to be read in the lower scale can be studied, and we come to the following conclusion:

1. For cinematic viscosities, which occur in practice and are 0.01 sq cm per sec for water at 20°C, the influence of the viscosity is negligible; both the ratio

$$\frac{r_m}{r_2} \text{ and } \frac{v_{\max}}{v_{\text{nonvis}}} \text{ approach one very quickly.}$$

This indicates that, with normal fluids, the tangential velocities behave similarly to nonviscous fluids for which vr = constant.

2. For very high cinematic viscosities the radius r_m, at which maximum tangential velocity $v_{\max}$ occurs, is 2.7 times (exactly $e = 2.718$) the outlet radius r_2, whereas this velocity for a special case where $r_2 = 6r_2$ is $\frac{1}{\ln r_1/r_2} = 0.558$ times the velocity of nonviscous flow at the radius r_m. The latter velocity can be calculated as: $v_1 r_1 = v_{\text{nonvis}} r_m$.

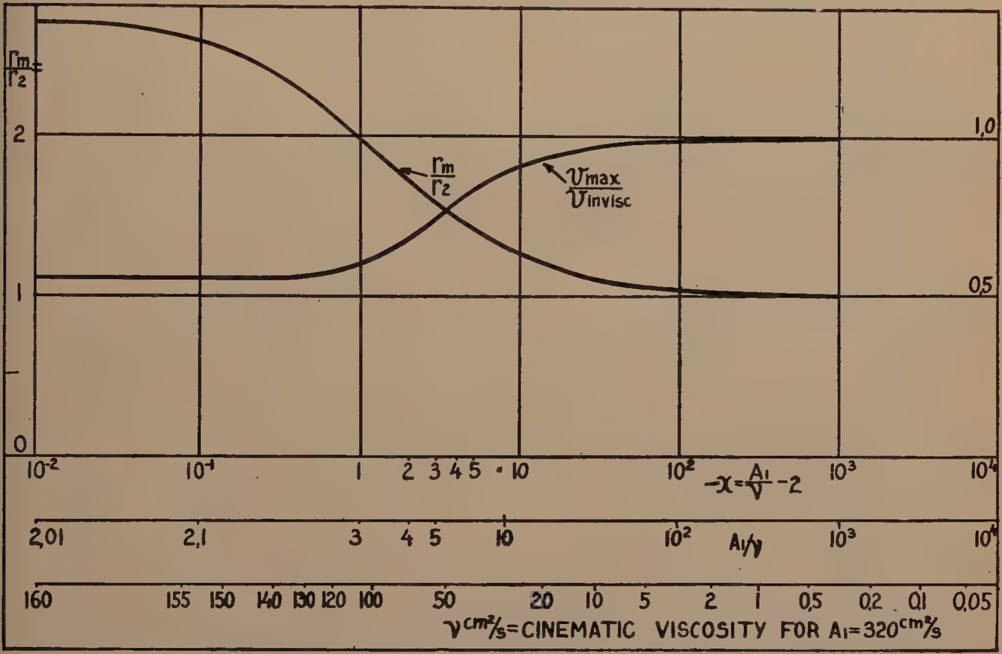


FIG 10—EFFECT OF CINEMATIC VISCOSITY ν ON RADIUS r_m OF MAXIMUM TANGENTIAL VELOCITY $v_{\max}$. AND ON RATIO $v_{\max.}/v_{\text{nonviscous}}$.
$v_{\text{nonvis.}}$ is maximum tangential velocity at r_m for ideal vortex motion.

VELOCITY OF PARTICLES IN MEDIA (WATER OR AIR) DUE TO GRAVITATIONAL OR CENTRIFUGAL FORCES

Gravitational Forces

The velocity of fall for particles in media has been thoroughly described in literature.[2,6] The laws governing the motion in viscous liquids can be divided into three ranges, which depend on the Reynolds number $\left(R_e = \frac{wd}{\nu}\right)$, where w is the velocity of fall of the particle, d its diameter, and ν the cinematic viscosity of the medium.

Range I—For small Reynolds numbers

($R_e < 1$), the well-known Stokes' law applies:

$$w = \frac{1}{18} g d^2 \frac{\gamma_s - \gamma}{\mu} \quad \text{Stokes} \qquad [20]$$

Where γ_s = the specific gravity of a sphere,
γ = the specific gravity of the fluid,
μ = the viscosity of the fluid.

Range II—As a rule, the motion of small particles in a heavy-liquid bath is not governed by Stokes' law but by the middle portion of a curve indicating the relation between R_e and the resistance factor C_w (ref. 2, Fig 1). This part can be approximated by Allen's formula:

$$w = \left(\frac{48}{30\gamma\sqrt{\nu}}\right)^{2/3} d(\gamma_s - \gamma)^{2/3} \qquad [21]$$

Range III—For increasing Reynolds numbers, c_w is a constant, and w is described by the law of Newton-Rittinger:

$$w = \frac{2}{3}\sqrt{3gd\frac{\gamma_s - \gamma}{\gamma c_w}} \qquad [22]$$

This equation, however, can be used for all three ranges, if the exact value of c_w is inserted. As, however, c_w depends on R_e, which again is proportional to w, it is a method of trial and error to compute the exact value in the equation. In order to eliminate the elaborate calculation of w in liquids or gases, F. Fontein, engineer of the Mining Research Station of the Staatsmijnen in Limburg, The Netherlands, has designed a very practical chart (Fig 11) by which it is possible to find at once the velocity of particles of varying density and sizes in liquids and gases of varying densities and viscosities.

By plotting $y = \log w$ against $x = \log \frac{d}{\nu} = \log \frac{d\gamma}{\mu}$ the Reynolds numbers $R_e = \frac{wd}{\nu}$, appear as straight lines under 45° because the sum of the abscissas and ordinates is a constant for R_e = constant:

$$x + y = \log\frac{d}{\nu} + \log w = \log\frac{wd}{\nu} = \log R_e$$
$$= \text{constant}$$

It can be proved easily that in this graph the Stokes and Newton-Rittinger laws also appear as straight lines, whereas the motion approximated by Allen's law must be drawn as a gradual transition of Range I of Stokes to Range III of Newton-Rittinger.

TABLE 4—*Effect of Cinematic Viscosity ν on Vortex Motion in a Cyclone*

ν Sq Cm per Sec	$\frac{A_1}{\nu}$	$\frac{A_1}{\nu} - 2 = -x$ [a]	$\frac{r_m}{r_2} = \left(\frac{1}{1-x}\right)^{\frac{1}{x}}$	$\frac{v_{max}}{v_{nonvisc}} = \frac{1 - \left(\frac{r_m}{r_2}\right)^x}{1 - \left(\frac{r_1}{r_2}\right)^x}$ [b]
160	2	0	2.718	0.55
160	2.001	10	2.71	0.557
160	2.01	10	2.704	0.557
153	2.1	10	2.594	0.554
128	2.5	0.5	2.52	0.563
107	3	1	2.0	0.60
80	4	2	1.73	9.69
64	5	3	1.59	0.75
53	6	4	1.50	0.80
29	11	9	1.29	0.90
27	12	10	1.271	0.909
3.13	102	10^2	1.047	0.99
0.32	1002	10^3	1.007	0.999
0.032	10002	10^4	1.001	0.9999

[a] For constant $A_1 = \frac{Q}{2\pi B}$ we have taken the same numerical value as the example on page 12:

$$A_1 = \frac{0.01}{2\pi 0.05} = 0.032 \text{ sq m per sec} = 320 \text{ sq cm per sec}$$

Q being 36 cu m per hr = 0.01 cu m per sec, A_1 must be expressed in the same dimensions as ν, thus in square centimeters per second.

[b] For our case we have figured with: $\frac{r_1}{r_2} = 6$

and $$\frac{v_{max}}{v_{nonvisc}} = \frac{v_{max}}{K/r_m} = \frac{\frac{1}{x-1}}{1-6^x}$$

The three ranges are now clearly indicated and divided by lines of constant R_e (Fig 11).

As mentioned before, not the diameter of the particle d but the logarithm of the expression

$$\frac{d}{\nu} = \frac{\text{diameter of particles}}{\text{cinematic viscosity}} = \frac{d\gamma}{\mu}$$
$$= \frac{\text{dia} \times \text{sp gr of liquids}}{\text{abs viscosity of liquids}}$$

is plotted on the abscissa.

From this value, a vertical line must be drawn until it intersects with the curved lines for constant $\mu\frac{\gamma_s - \gamma}{\gamma^2}$

At the point of intersection, the value for w can be read on the vertical axis.

On the abscissa, as well as on the w-axis, two scales are shown, one for liquids and one for gases. The oblique and curved lines have also scales for gases and liquids, so that the graph can be used to compute the velocity of fall of particles in liquids as well as in gases.

The left part of the graph, limited by the thick straight line under 45° for $R_e = 10^{-1}$, covers Range I of Stokes' law, where the velocity of fall is proportional to d^2.

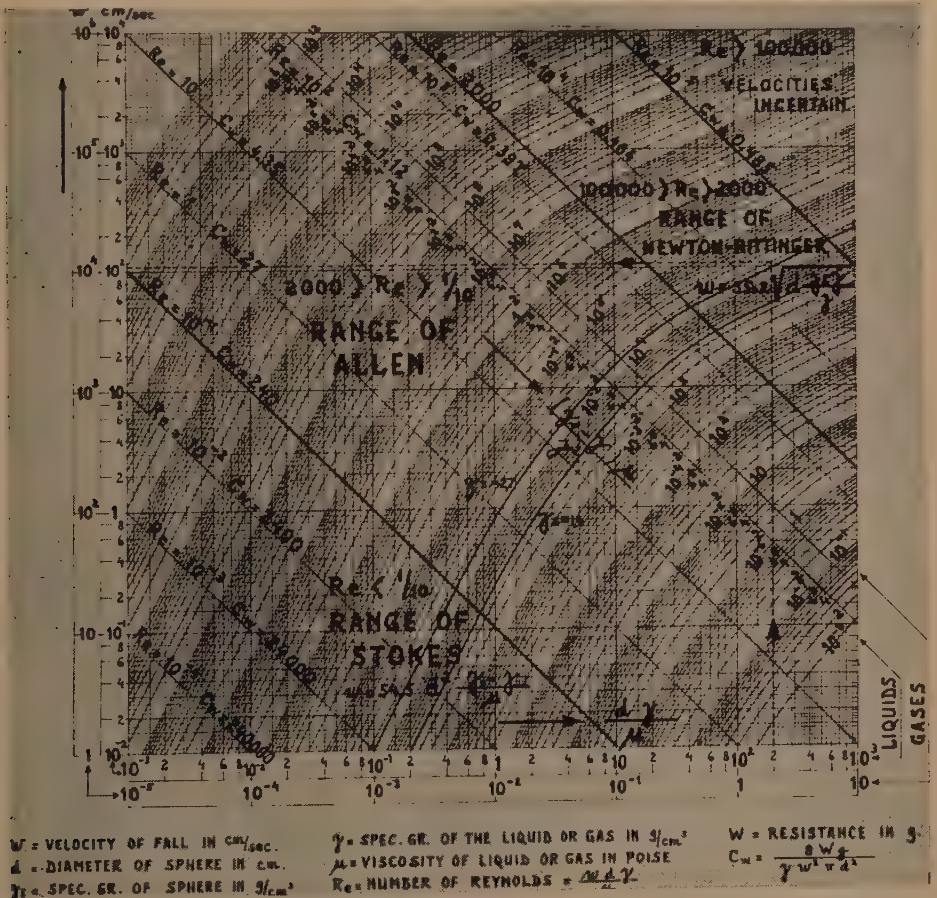


FIG 11—FONTEIN'S CHART FOR COMPUTING VELOCITIES OF FALL OF SPHERICAL PARTICLES IN LIQUIDS AND GASES.

Example: Required is velocity of fall w in water of 20°C of ball with diameter $d = 2$ cm. Specific gravity of water $\gamma = 1$ gram per cu cm.

Abscissa viscosity of water $\mu = 0.01$ poise. Specific gravity $\gamma_s = 3$ grams per cu cm.

Calculate $\frac{d\gamma}{\mu} = 200$ and $\frac{\gamma_s - \gamma}{\gamma^2}\mu = 2 \times 10^{-2}$. From value 200 on abscissa follow arrow to intersection of curve for which $\frac{\gamma_s - \gamma}{\gamma^2}\mu = 2 \times 10^{-2}$ and find $w = 110$ cm per sec. The scale for liquids must be used.

The following Range II, between $R_e = 10^{-1}$ and $R_e = 2000$, is the transition from Stokes' law to the Newton-Rittinger law, and the velocity is approximately described by Allen's law, where w is proportional to d.

Range III, between $R_2 = 2000$ and $R_e = 10^5$, can be described by the law of Newton-Rittinger, where w is proportional to the square root of d.

The range beyond $R_e = 10^5$ does not occur in washery practice, so the calculation of w at these high Reynolds numbers will not be discussed in this paper.

If the particles are not spherical, but angular or flat, as for coal or shale particles, the velocity of fall w is smaller, and we must introduce a factor f by which w for spherical particles should be multiplied.

$$w_{\text{coal or shale particles}} = f \times w_{\text{spherical particles}}$$

As can be foreseen, this factor f again depends on the Reynolds number. Assuming a ratio

$$\frac{\text{thickness}}{\text{diameter}} = \frac{a}{d} = 0.7 \text{ for coal}$$

and

$$\frac{a}{d} = 0.4 \text{ for shale}$$

the factor f can be found with Table 5.

TABLE 5—*Factor f*

Velocity of Fall for Spherical Particles Must be Multiplied by f in Order to Obtain Velocities for Coal and Shale Particles

Diameter of Particle in Water, Cm.	Reynolds Number, R_e	Factor f		Range
		For Coal, $\gamma_s < 1.3$	For Shale $\gamma_s > 2.0$	
<0.01	0–1	0.96	0.56	I (Stokes)
About 0.14	100–200	0.70	0.50	II (Allen)
>0.4	10^3–10^5	0.57	0.45	III (Newton-Ritt.)

The factors f were determined by Fontein in measuring the velocities of fall of small coal and shale particles of 1 to 2 mm, the exact dimensions of which were ascertained before the tests.

Centrifugal Forces

The only difference in the equations for the velocities of fall due to gravity and the velocities due to the centrifugal force is that for the latter the acceleration of gravity g must be replaced by the acceleration of centrifugal force $\frac{v^2}{r}$.

In the Stokes' equation 20, w is proportional to g, thus the velocity in the centrifugal field is proportional to $\frac{v^2}{r}$, and the factor c, by which the velocities of fall must be multiplied to obtain the velocities in the field of centrifugal force, is:

$$c_{\text{Stokes}} = \frac{v^2}{rg}$$

In Allen's Eq 21, w is proportional to $g^{2/3}$; thus,

$$c_{\text{Allen}} = \left(\frac{v^2}{rg}\right)^{2/3}$$

In the Newton-Rittinger Eq 22, w is proportional to $\sqrt{g}$; thus,

$$c_{\text{Newton-Rittinger}} = \sqrt{\frac{v^2}{rg}}$$

In the graph of Fig 11, the increased acceleration due to the centrifugal forces can be taken care of in multiplying the parameter $\frac{(\gamma_s - \gamma)\,\mu}{\gamma^2}$ by the factor $\frac{v^2}{rg}$, irrespective of the range.

As an example, we assume in Fig 12 that $\frac{v^2}{rg} = 10$. The parameter $\frac{\gamma_s - \gamma}{\gamma^2}\,\mu$ must be increased by 10 to obtain the required velocity $w_{o,f}$ in the centrifugal field for the same value, $d\,\frac{\gamma}{\mu}$.

We remark that the centrifugal force

has a relatively larger effect on the small "Stokes" particles than on the somewhat larger "Allen" particles.

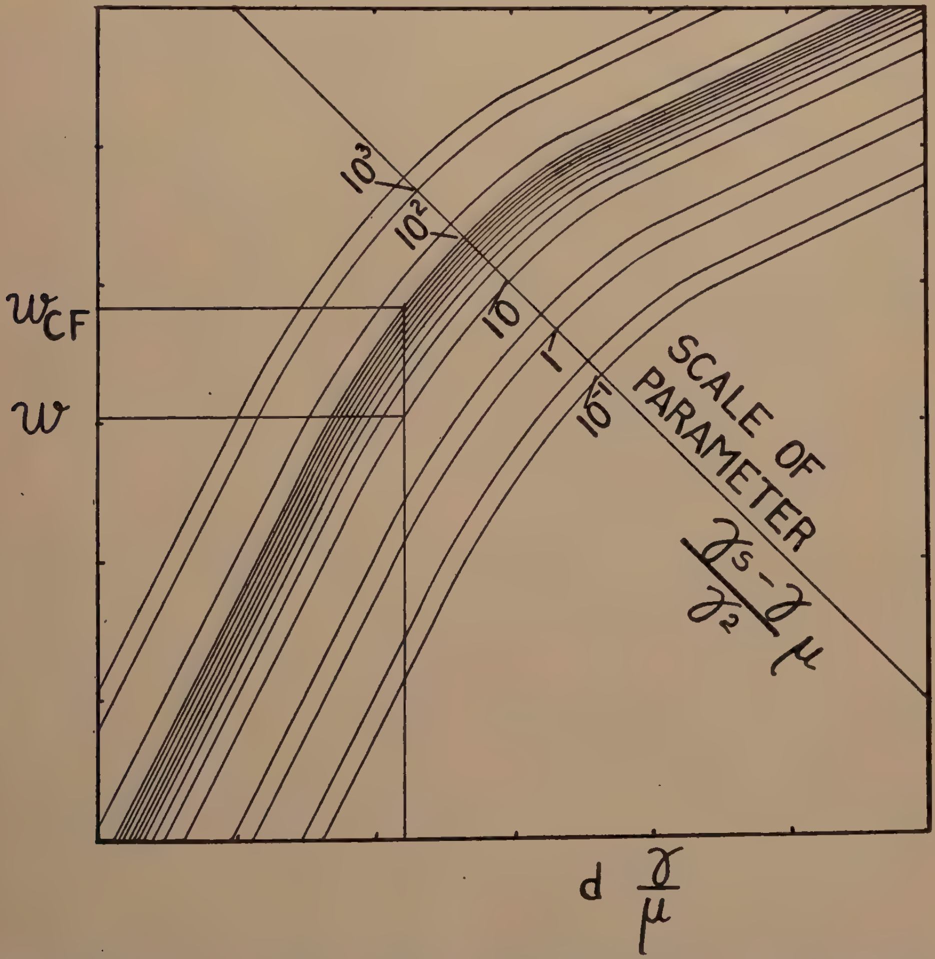


Fig 12—Method of computing velocity w_{cf} of spherical particles in centrifugal field by multiplying the parameter $\frac{\gamma_s - \gamma}{\gamma^2} \mu$ of graph 11 with $\frac{v^2}{rg}$,

w = velocity of fall,
w_{cf} = velocity in centrifugal field.

The graph, Fig 11, is specially designed for Range II of Allen, as the velocities of Ranges I (Stokes) and III (Newton-Rittinger) can be easily calculated by means of the Eqs 20 and 22. It should be remarked that Allen's law (21) should also give a straight line in the logarithmic diagram of Fig 11, but the curved lines give a better approximation to the true velocities of fall computed with help of the exactly measured c_w values (ref. 2, Fig 1) than Equation 21. Another observation is that all velocities w are ultimate velocities of isolated spheres in a theoretically unlimited medium. For coal or shale particles moving amidst other particles, the laws for the ultimate velocities are liable to considerable corrections.[6]

SUMMARY

The principle and behavior of cyclone thickeners and washers are described and flowsheets showing the position of the cyclones in heavy-medium washeries are included.

A method of error curves, to judge the effect of separating apparatus, is developed and specialized as well for the cyclone thickeners separating small particles into coarser and finer fractions as for cyclone washers separating particles of various specific gravities into heavier and lighter fractions.

An example for the cyclone thickener is given, showing that the "separating size" where 50 pct of a certain fraction is caught is situated at a grain size of about 35μ.

An example for the cyclone washer treating a middling product of 44.5 pct ash shows that it is possible to wash particles down to 0.5 mm with a very high approximation to theoretical separation, whereas a washing effect down to 0.1 mm could be demonstrated.

The theory of the flow in a cyclone is discussed under the assumption that it can be compared with an ideal sink in a nonviscous liquid. With the help of the Navier-Stokes equations in polar coordinates, a first approximation considering the effect of viscosity on the maximum tangential velocity near the cyclone core can be obtained.

For the velocities of small particles in media under the influence of gravitational or centrifugal forces, a new chart is designed, which gives the solution for a certain range of particle sizes and velocities that cannot be calculated by an exact equation.

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APPENDIX

Method to Calculate Partial Recoveries (Error Curve)

Let us suppose A (Fig 13) to be a separating vessel of arbitrary construction having a feed f and a discharge of the separated products c (coal) and d (dirt). The feed f may consist of particles, which can be distinguished by a certain physical property such as size, specific gravity, sulphur content, surface tension, and so on. We further assume that the physical property can be evaluated by a number, such as the size by the diameter, and we then can divide the feed as well as the product c and d in fractions limited by these numbers. Let us assume that these numbers actually denominate the size of the particles and that we are able to ascertain by experiment the size fractions of c and d, but also the ratio of weight of the solids $\frac{c}{d}$.

As $f = c + d$, we also know $\frac{c}{f} = \frac{c}{c+d}$ and

$$\frac{d}{f} = \frac{d}{c+d}.$$

We then can calculate for each size-fraction the recovery in one of the products c or d. For instance the recovery r_x of a certain fraction in the product c is given by:

$$r_x = \frac{\text{the weight of the fraction in } c}{\text{the weight of the fraction in } c + \text{the weight of the fraction in } d}$$

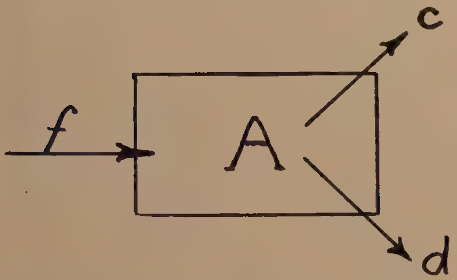

FIG 13—SHOWING SEPARATING VESSEL A WITH FEED f AND PRODUCTS c (COAL) AND d (DIRT).

As a rule we will express the weights of the fractions in percentages of the products. Thus x_c may denominate the percentage of the fraction in question available in the product c. Similarly x_d the fraction in d. We now have

$$r_x = \frac{x_c \dfrac{c}{c+d}}{x_c \dfrac{c}{c+d} + x_d \dfrac{d}{c+d}} = \frac{x_c c}{x_c c + x_d d}$$

$$= \frac{x_c \dfrac{c}{d}}{x_c \dfrac{c}{d} + x_d}$$

As the latter expression only contains the ratio in weight of the products c and d and the fractions of a certain size x_c and x_d expressed in percentages of the products c and d, respectively, a measurement of $\frac{c}{d}$ and a size analysis of the products will suffice to compute the recoveries or efficiencies of the individual size fractions.

If $\frac{c}{d}$ is not directly measured $\frac{c}{d}$ can be calculated from one of the size fractions in the feed f and in the products c and d according to the following formula:

$$\frac{c}{d} = \frac{x_f - x_d}{x_c - x_f}$$

x_f standing for the size fraction in the feed.

The Cyclone as a Thickener of Coal Slurry*

By H. F. Yancey† and M. R. Geer,† Members AIME

(Cincinnati Meeting, October 1947)

With the exception of pneumatic processes and a few special beneficiation methods of comparatively limited application, all mechanical coal-cleaning and mineral-dressing processes involve the admixture of solid particles with a liquid, generally water. Upon completion of the beneficiation process the solids must be reclaimed from the water. With coarse particles separation of solids from water is accomplished readily by the simple expedient of screening, but recovery of fine solids presents a much more difficult problem. Thickening and dewatering are practiced so generally, both within the mineral industry and elsewhere, that they have been the subject of study through the years and a number of processes for making the solid-liquid separation have been developed. Dorr-type thickeners have been standard equipment for many years, and more recently driers employing centrifugal force have gained in popularity.[1]

A cyclone device employed to thicken the loess suspension is a novel feature of the loess heavy-medium coal-cleaning process developed by the Netherlands State Mines in the thirties. The cyclone thickener was first mentioned by Driessen[2] in 1939 in a paper describing the loess process and has been the subject of two more recent reports by the same author.[3] The cyclone does not differ greatly from cyclone dust collectors used in this country for years, except that it is operated wet rather than dry. Advantages of the cyclone over other thickening devices are its simplicity of construction, high capacity, and the fact that, having no moving parts, it is simple and inexpensive to operate.

The Bureau of Mines became interested in the cyclone after one of the present authors observed it in operation in the Netherlands in 1945. The first experimental work with the cyclone conducted by the Bureau of Mines was an investigation of the unit functioning as a heavy-medium cleaner for fine coal—a use for the cyclone that was conceived long after it had been perfected as a thickener. The results of this investigation were published last year.[4] The present investigation deals exclusively with operation of the cyclone as a thickener.

Object and Scope of Investigation

Although the cyclone thickener has been dealt with in three reports by Driessen, its use as a thickener was not the principal subject of any of these reports and therefore neither the cyclone itself nor its operation was described completely. The design of the cyclone was described only in a very general way, only limited operating results were given, and virtually nothing was disclosed about the influence of the various possible operating variables. Moreover, the Netherlanders apparently had employed the cyclone exclusively for thickening loess and had not employed it on coal slurries. One English colliery has used a cyclone to thicken coal slurry,[5] and as it appeared that

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* This report gives the results of work done under a cooperative agreement between the U. S. Department of the Interior, Bureau of Mines, and the University of Washington. Published by permission of the Director, Bureau of Mines. Manuscript received at the office of the Institute October 30, 1947. Issued as TP 2351 in Coal Technology, February 1948.

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[1] References are at the end of the paper.

the Netherlands-type cyclone might have wide application as a thickener for coal slurries in this country, the Bureau of Mines launched an investigation to provide information about construction and operation. The investigation has been in progress for about six months and it has become apparent that considerable time will be required to study completely all aspects of cyclone design and operation. Therefore, since the results obtained thus far have been so promising, it was considered advisable to publish them now for the benefit of the coal industry rather than to await completion of the investigation.

The cyclone used so far is a unit 5 in. in diam having a capacity of 40 to 50 gpm. This unit probably is smaller than normally would be used in a plant but serves admirably for testing purposes because the small capacity permits a large number of tests to be made in minimum time—a desirable feature when investigating a device about which so little is known.

Most of the investigation was conducted in the laboratory where control over all operating factors is more easily established and where samples can be examined more readily. Once satisfactory operation of the cyclone was achieved, however, it was installed temporarily in two coal washeries where performance under plant conditions could be observed.

DESCRIPTION OF EQUIPMENT AND TESTING PROCEDURE

Fig 1 illustrates the construction of one of the cyclones used in this investigation. The conical section, which is the heart of the cyclone, is 5 in. max diam, 1 in. min diam, and 11.3-in. high; the total inclosed angle of the cone is 20°. Immediately above the conical section is a cylindrical ring 1⅜ in. high and 5 in. in diam, through which the feed tube enters the cyclone. The feed tube, which tapers from 1½-in. to 1¹⁄₁₆-in. diam in a length of 7½ in., enters the cyclone on a tangent with the cylindrical ring and is inclined downward at an angle of 5° from the horizontal. Surmounting the cylindrical ring is a removable orifice plate fitted with a tube extending down into the cyclone. The chamber above the orifice plate serves to collect and discharge the overflow water through a 1½-in. pipe at one side of the chamber. A sleeve within the overflow chamber holds the orifice plate down by means of pressure from the screw cap.

In operation the slurry of water and fine coal is pumped through the tangential feed nozzle and thereby caused to rotate or whirl. Under the influence of the centrifugal force developed by the rotary motion, the solid particles are thrown to the wall of the conical section and forced downward along the wall until they pass out of the cyclone through the underflow or bottom opening. The water, largely freed of solid particles, overflows through the central opening in the orifice plate at the top and leaves the cyclone by way of the overflow chamber.

In designing the cyclone means were provided for varying the size of the various openings. The feed nozzle can be varied in size by inserting sleeves, the size and type of overflow opening can be varied by means of interchangeable orifice plates, and the underflow opening can be varied in size by screwing plugs having different openings into the bottom section of the cyclone. Also, the cyclone was constructed so that interchangeable conical sections of various slopes can be used.

Fig 2 is a photograph of the laboratory installation. The cyclone is mounted over a 50-gal conical tank which serves as a collecting sump for the feed pump. Thus the cyclone is in a closed circuit, both overflow water and underflow coal falling directly into the conical tank and from there being recirculated through the cyclone. The centrifugal pump is provided with a variable-speed drive capable of developing from 0 to 50 psi pressure at the feed nozzle of the cyclone.

In making a test, slurry is introduced into the conical tank and the cyclone is operated until the optimum combination of operating adjustments, as judged visually, is obtained. Time samples of both overflow and underflow products are then collected simultaneously for examination. For most tests the percentage of solids in both samples is determined and the solids are screened to determine particle size. In some cases the solids are analyzed for ash content. The composition of the feed to the cyclone is calculated from the composition of the two products.

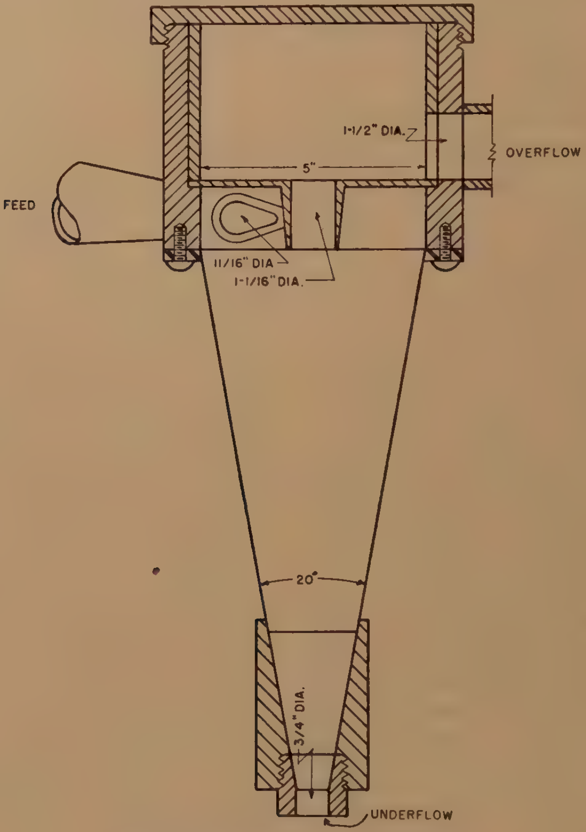

FIG 1—CONSTRUCTION OF CYCLONE.

Slurries Tested

In most of the preliminary work of determining the basic operating characteristics of the cyclone an artificial slurry was used. Coal from the Big bed in the Roslyn No. 3 mine, Ronald, Wash., was crushed to the desired top size and mixed with water to obtain a slurry having the pulp ratio required for a particular test. The top particle size employed in these tests ranged from 6- to 65-mesh, and the slurries varied from 0.3 to 39.0 pct solids.

Following the preliminary work the cyclone was installed in the central cleaning plant of the Northwestern Improvement Co., Roslyn, Wash., which handles coal from the company's three mines in that district. While installed in this plant the cyclone was operated on the effluent from a centrifugal drier—a slurry with about 20 pct solids, having a top size of 28-mesh, and containing about 50 pct of material finer than 200-mesh. Tests were made also on the overflow product of a large, conical settling tank of the type frequently used at washeries to provide clarified water for recirculation in the plant. This overflow product contained 8 pct solids, about 80 pct of which was finer than 200-mesh.

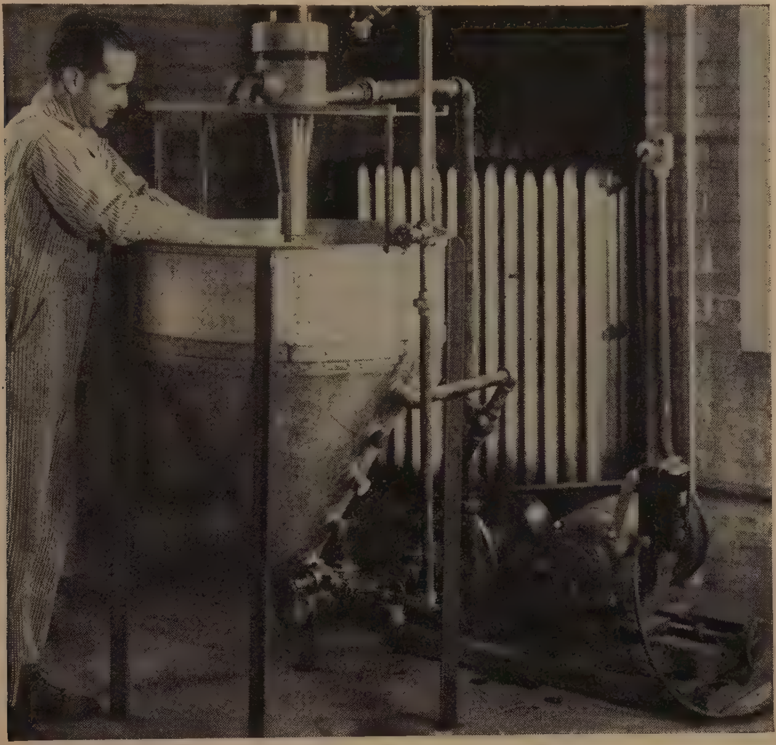

FIG 2—CYCLONE TEST UNIT ARRANGED FOR CLOSED-CIRCUIT OPERATION.

The cyclone was subsequently installed in the washery of the Springbrook Mining Co., Renton, Wash., where it was used to recover solids from the slurry wasted from a settling tank. This slurry contained 4 pct solids having a top size of 35-mesh and averaging about 50 pct finer than 200-mesh.

FORMS OF CYCLONE UNDERFLOW PRODUCT

The form in which the underflow product emerges from the cyclone provides the best means of judging cyclone operation. On starting the cyclone, if the sizes of the overflow and underflow openings are properly adjusted for the feed rate and the character of the slurry being treated, the underflow product will emerge from the bottom of the cyclone in the form of a strong spray or vortex of water and coal. If, however, the underflow opening is closed momentarily to permit a bed of coal to accumulate in the base of the cyclone, the vortex is replaced by a rotating or twisting solid rod of coal that resembles sausages coming from a mill. The "sausages" form, break off and drop, and then re-form. Fig 3 is a photograph of sausages that were produced from 35-mesh to 0 coal; they contained 65 pct solids as they came from the cyclone and retained their shape perfectly on drying.

Table 1 presents the results of two tests made on the same slurry with identical cyclone adjustments; in test 82 the underflow was a vortex, while for test 81 the underflow was converted to sausage form. In the test made with vortex operation 12.3 pct of the water present in the cyclone feed entered the underflow product, and the underflow was thus diluted to 39.5 pct solids. In the test representing a sausage-type underflow only 5.3 pct of the water present in the feed entered the underflow product, and the underflow contained 59.4 pct solids. A sausage-type underflow always contains more solids than does a vortex underflow—the magnitude of the difference depending upon the concentration of solids in the cyclone feed. In treating dilute slurries the sausage-type underflow may contain as much as five times the concentration of solids obtained in a vortex underflow, while in treating thicker slurries the difference between the two types of underflow product is much less pronounced.

The figures at the bottom of Table 1 show what percentage of each size fraction of the solids entered the overflow water and what percentage was recovered in the underflow product. The only noticeable difference in the recovery of solids obtained in operating the cyclone with sausage-type and vortex underflows occurs in the material finer than 200-mesh. There was virtually no loss of solids coarser than 200-mesh in either test, but the material finer than this size was recovered more efficiently when the cyclone was operated with a vortex underflow. For example, with a sausage-type underflow only 20 pct of the material finer than 400-mesh was recovered, while with vortex operation the recovery in this size was 43.9 pct. Thus, in treating a slurry containing a high percentage of material finer than 200-mesh, vortex operation will provide a higher total recovery of solids, but in treating a slurry containing only a moderate amount of extreme fines the recovery obtained with the two types of underflows is substantially the same.

Even when conditions for a sausage-type underflow are ideal, the cyclone will continue to operate indefinitely with a vortex

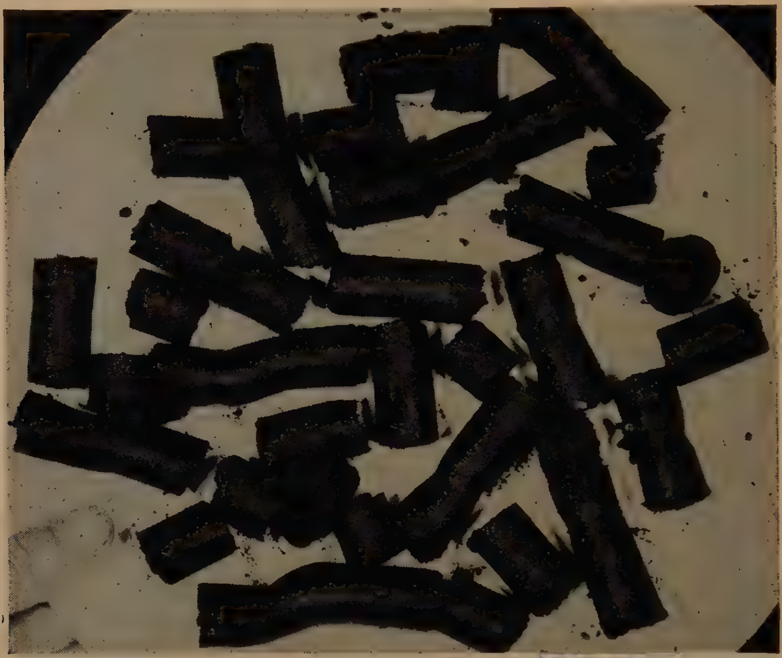

Fig 3—Unusually dry cyclone underflow in the form of "sausages,"

unless the underflow opening is closed momentarily to permit a bed of coal to accumulate. Whether or not a sausage-type underflow can be formed depends upon several factors. First, there must be sufficient coarse solids in the feed to form and maintain a bed in the bottom of the cyclone. Second, the cyclone must be operated at a feed-nozzle pressure high enough to insure a speed of rotation that will throw the solids out of suspension. And third, the size of the underflow opening must be adjusted properly in relation to the amount of material it must pass. Once these conditions are established, however, the sausage-type underflow will persist despite wide fluctuations in the character of the cyclone feed—the concentration of solids in the feed may change, the size composition of these solids may vary, or the quantity of slurry entering the cyclone may fluctuate without destroying the sausage-form of the underflow. The rate at which the sausages emerge from the cyclone automatically adjusts itself to the rate at which solids are added to the bed in the bottom section of the cone.

TABLE 1—*Comparison of Cyclone Operation with Vortex- and Sausage-type Underflows*

	Test 82, Vortex-type Underflow			Test 81, Sausage-type Underflow		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Solids, pct	8.0	0.8	39.5	8.0	1.0	59.4
Distribution of total solids, pct	100.0	7.7	92.3	100.0	10.8	89.2
Distribution of water, pct	100.0	87.7	12.3	100.0	94.7	5.3
Screen analysis of solids, pct						
35 to 48	24.4	0.0	26.4	23.8	0.0	26.7
48 to 65	12.7	0.0	13.8	20.0	0.0	22.4
65 to 100	23.5	0.1	25.5	19.2	0.1	21.5
100 to 150	9.1	0.0	9.8	9.4	0.0	10.5
150 to 200	9.6	0.4	10.4	8.6	0.8	9.6
200 to 270	2.6	0.2	2.8	2.6	1.5	2.7
270 to 400	4.6	1.1	4.9	4.4	8.2	3.9
Under 400	13.5	98.2	6.4	12.0	89.4	2.7
Distribution of solids by sizes, pct						
35 to 48	100.0	0.0	100.0	100.0	0.0	100.0
48 to 65	100.0	0.0	100.0	100.0	0.0	100.0
65 to 100	100.0	0.0	100.0	100.0	0.0	100.0
100 to 150	100.0	0.0	100.0	100.0	0.0	100.0
150 to 200	100.0	0.3	99.7	100.0	1.0	99.0
200 to 270	100.0	0.8	99.2	100.0	6.2	93.8
270 to 400	100.0	1.7	98.3	100.0	20.4	79.6
Under 400	100.0	56.1	43.9	100.0	80.0	20.0

In adjusting the cyclone, if the underflow opening is excessively large for the slurry being treated, too much water will reach the bottom and a sausage-type underflow cannot be obtained. If, on the other hand, the underflow opening is too small to provide egress for the solids as they are separated from the water, the cyclone will become overloaded. When overloaded with solids the speed of rotation within the cyclone is decreased, less centrifugal force is developed, and consequently the separation of solids from water becomes inefficient; under this condition an excessive amount of coarse solids will enter the overflow product. The overloaded condition is reflected in the form of the underflow product, which loses the twist that characterizes a good sausage-type underflow and emerges from the cyclone as a lazy, straight, solid stream containing more water than that in a sausage-type underflow.

When the cyclone is operating with a sausage-type underflow the water passing up through the overflow orifice rotates with sufficient speed to create a vacuum in the area just under the center of the cap on the overflow chamber. In fact, if this vacuum is broken by opening a petcock in the center of the cap the sausage-type underflow immediately breaks out into a vortex. When the cyclone is overloaded with solids, however, the water does not rotate fast enough to create a vacuum in the overflow chamber, and the absence of the vacuum is an additional means of diagnosing the overloaded condition.

At this stage of the investigation it is impossible to specify with certainty how much coarse material a slurry must contain to give a sausage-type underflow from the cyclone. In general it appears, however, that the solids in the slurry should contain at least 35 or 40 pct of material coarser than 200-mesh, depending somewhat on the concentration of solids in the slurry, Thicker slurries can contain more extreme fines than is permissible in dilute slurries.

Whether the cyclone should be operated with a vortex or a sausage-type underflow is entirely a question of the performance desired. If maximum recovery of solids, including the sizes finer than 200-mesh, is the principal consideration, vortex operation should be employed. If, however, the maximum concentration of solids in the underflow product is required, a sausage-type underflow should be used. Operation of the cyclone with a sausage-type underflow is best also if it is desired to recover the coarser solids but reject the extreme fines, for example, in recovering coal from a slurry containing considerable clay.

DESIGN FACTORS

At the outset of the investigation so little was known about the construction of the cyclone that much of the preliminary work involved investigation of elements of cyclone design. The relative size of the feed-nozzle, overflow, and underflow openings, the type of overflow opening, and a number of other design factors had to be investigated before the performance of the cyclone could be evaluated.

Influence of Feed-nozzle Pressure on Cyclone Performance

The feed-nozzle pressure employed in cyclone operation influences the capacity of the cyclone, and, since it determines the speed of rotation and thus the centrifugal force developed, influences the separation of solids from water. With the cyclones used in this investigation sausage-type underflows could not be obtained at feed-nozzle pressures of much less than 15 psi and consequently test work was confined to the range of 15 to 80 psi, the maximum pressure obtainable.

Table 2 presents the results of cyclone tests made on the same slurry at 15 and 50 psi, all other operating adjustments remaining the same. In test 36, made at 50 psi, the cyclone handled 38 gpm or about 50 pct more water than the throughput obtained in test 35 at 15 psi. The increase in capacity caused by increased nozzle pressure was thus 83 pct of the increase that would be calculated by assuming the flow to be proportional to the square root of the pressure increase. The fact that the solids capacity of the cyclone—400 lb per hr—was the same in both tests is merely a coincidence; if the feed solids had been the same in both tests the throughput of solids in the 50 psi trial would have been 50 pct greater than that obtained at 15 psi.

The underflow product obtained at 50 psi contained 60.9 pct solids in comparison with 57.0 pct solids in the underflow product at 15 psi. In general the underflow is somewhat drier at higher feed pressures but the advantage gained in this respect seldom is more than a few per cent.

Inspection of the figures in Table 2 showing the distribution of solids by sizes indicates that increasing the feed pressure from 15 to 50 psi had no appreciable effect on the recovery of any size fraction in the underflow product. In neither test was there a substantial loss of any material coarser than 200-mesh, and the treatment of material finer than this size was about the same at both feed pressures. It would be natural to expect that higher feed pressure, resulting in greater centrifugal force, would provide better recovery of the finest solids. However, higher feed pressure also increases the load of material being treated in the cyclone—a factor which militates against more efficient clarification. If the feed nozzle were decreased in size so that the load of material in the cyclone was the same at higher feed pressure, the effect of greater centrifugal force in removing very fine solids from the water would become more apparent.

TABLE 2—*Comparison of Cyclone Operation at 15 and 50 psi Feed-nozzle Pressure*

	Test 35, 15 psi			Test 36, 50 psi		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Throughput, water, gpm	25.1			37.9		
Throughput, solids, lb per hr	400.			400.		
Solids, pct	3.7	0.1	57.0	2.2	0.1	60.9
Distribution of solids, pct	100.0	2.6	97.4	100.0	5.9	94.1
Screen analysis, pct						
35 to 48	39.7	0.2	40.7	24.2	0.3	25.7
48 to 100	44.3	4.8	45.4	47.3	0.5	50.2
100 to 200	10.8	8.1	10.9	16.7	8.9	17.2
Under 200	5.2	86.9	3.0	11.8	90.3	6.9
Distribution of solids by sizes, pct						
35 to 48	100.0	0.0	100.0	100.0	0.1	99.9
48 to 100	100.0	0.3	99.7	100.0	0.1	99.9
100 to 200	100.0	1.9	98.1	100.0	3.2	96.8
Under 200	100.0	43.6	56.2	100.0	45.1	54.9

A number of other companion tests have been made on various slurries at 15 and 50 psi, and in every case the influence of higher feed pressure was substantially the same as that illustrated in tests 35 and 36, that is, the capacity of the cyclone was increased about 50 pct, the underflow product was slightly drier, but the recovery of coal in the underflow product was practically the same.

Comparison of Types of Overflow-orifice Plates

In all of the initial cyclone tests the type of overflow-orifice plate employed was essentially like the one shown in Fig 1 except that the plate was inverted with the tube extending up into the overflow chamber instead of down into the cyclone. It was thought that the upward-pointing tube was necessary to prevent water in the overflow chamber from exerting back pressure on the cyclone. Actually, the upward-tube orifice was not only unnecessary but definitely inferior to the type of downward-tube orifice shown in Fig 1, as demonstrated by the results of comparable tests of the two types presented in Table 3. With the upward-tube orifice, represented by test 39, 2.3 pct of the solids coarser than 48-mesh entered the overflow water; similarly, 7.8 pct of the 48- to 100-mesh solids and 11.1 pct of the 100- to 200-mesh solids were carried out with the overflow water. In contrast, test 38 shows there was virtually no loss of solids coarser than 200-mesh with the downward-tube orifice. The downward-tube orifice also recovered 6.2 pct more of the solids finer than 200-mesh.

A simple, thin-plate, sharp-edge orifice having no tube projecting in either direction also was tried, but it likewise proved distinctly inferior to the downward-tube type, as demonstrated by the test results presented in Table 4. With the thin-plate orifice a substantial proportion of the material coarser than 100-mesh and 17.2 pct of the 100- to 200-mesh solids were carried in the overflow water, while with the downward-tube orifice the loss of solids became appreciable only at the 100- to 200-mesh size.

TABLE 3—*Comparison of Upward-tube and Downward-tube Orifice Plates*

	Test 38, Downward-tube Orifice			Test 39, Upward-tube Orifice		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Throughput, water, gpm	39.2			39.4		
Throughput, solids, lb per hr	2000.			1200.		
Solids, pct	10.2	0.6	62.2	6.1	1.0	61.2
Distribution of solids, pct	100.0	5.5	94.5	100.0	14.9	85.1
Screen analysis, pct						
35 to 48	33.9	0.1	35.9	24.3	3.7	27.9
48 to 100	42.3	0.1	44.7	42.6	22.3	46.2
100 to 200	12.5	0.7	13.2	16.1	12.0	16.8
Under 200	11.3	99.1	6.2	17.0	62.0	9.1
Distribution of solids by sizes, pct						
35 to 48	100.0	0.0	100.0	100.0	2.3	97.7
48 to 100	100.0	0.0	100.0	100.0	7.8	92.2
100 to 200	100.0	0.3	99.7	100.0	11.1	88.9
Under 200	100.0	48.2	51.8	100.0	54.4	45.6

TABLE 4—*Comparison of Downward-tube and Thin-plate Orifice Plates*

	Test 36, Downward-tube Orifice			Test 37, Thin-plate Orifice		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Throughput, water, gpm	37.9			39.7		
Throughput, solids, lb per hr	400.			400.		
Solids, pct	2.2	0.1	60.9	2.2	0.4	61.4
Distribution of solids, pct	100.0	5.9	94.1	100.0	15.5	84.5
Screen analysis, pct						
35 to 48	24.2	0.3	25.7	25.0	3.8	28.9
48 to 100	47.3	0.5	50.2	44.9	27.2	48.1
100 to 200	16.7	8.9	17.2	16.2	18.0	15.9
Under 200	11.8	90.3	6.9	13.9	51.0	7.1
Distribution of solids by sizes, pct						
35 to 48	100.0	0.1	99.9	100.0	2.4	97.6
48 to 100	100.0	0.1	99.9	100.0	9.4	90.6
100 to 200	100.0	3.2	96.8	100.0	17.2	82.8
Under 200	100.0	45.1	54.9	100.0	56.9	43.1

The superiority of the downward-tube orifice can apparently be ascribed to the fact that in the cylindrical section at the top of the cyclone where the feed enters, the separation between coal and water is incomplete—that is, some of the coal has not yet been forced to the wall; consequently overflow water withdrawn at this elevation contains some coarse coal. Clarification of the water becomes more complete as the slurry flows downward in the cyclone and the solid particles have had more time to be forced outward to the wall of the cone. The downward-tube orifices used thus far have removed the overflow product from an elevation corresponding to the juncture of the cylindrical and conical sections of the cyclone. While an overflow product withdrawn at this elevation is distinctly superior to one withdrawn at the top of the cyclone, the experiments are not yet sufficiently complete to demonstrate that this is necessarily the best point to take off the overflow water—still longer tubes have not been tried.

Discussion of Types of Overflow Launders

Strictly speaking, the orifice plate constitutes the top of the cyclone proper; all of the separation between coal and water takes place below the orifice plate, and the portion of the cyclone above this level is merely a chamber or launder for collecting the water as it issues from the orifice. The design of this chamber is of some importance, however, as it influences the back pressure on the orifice. The type of chamber illustrated in Fig 1 appears to be entirely top portion of the cyclone; the annular bottom of the chamber has a drainage slope toward the discharge opening, which is located in the bottom. An overflow chamber of this type was tried out in comparison with the type illustrated in Fig 1 but no appreciable difference in the performance of the cyclone could be detected.

TABLE 5—*Comparison of 10 and 20° Cone Sections, with Cyclone Operating on Slurry of Low Solids Content*

	Test 53—10° Cone			Test 54—20° Cone		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Cyclone openings, in.	1 1/16	1 1/4	7/8	1 1/16	1 1/4	7/8
Feed-nozzle pressure, psi	15			15		
Throughput, water, gpm	34.1			28.4		
Throughput, solids, lb per hr	400.			400.		
Solids, pct	2.3	0.1	58.2	2.8	0.4	57.1
Distribution of solids, pct	100.0	5.2	94.8	100.0	13.3	86.7
Screen analysis, pct						
35 to 48	22.7	0.1	24.0	21.5	0.2	24.8
48 to 65	26.2	0.1	27.6	27.1	0.7	31.2
65 to 100	21.4	0.2	22.6	18.1	2.1	20.5
100 to 150	10.3	0.3	10.8	10.5	4.8	11.4
150 to 200	9.8	1.7	10.2	8.6	10.8	8.3
Under 200	9.6	97.6	4.8	14.2	81.4	3.8
Distribution of solids by sizes, pct						
35 to 48	100.0	0.0	100.0	100.0	0.1	99.9
48 to 65	100.0	0.0	100.0	100.0	0.3	99.7
65 to 100	100.0	0.0	100.0	100.0	1.6	98.4
100 to 150	100.0	0.2	99.8	100.0	6.1	93.9
150 to 200	100.0	0.9	99.1	100.0	16.7	83.3
Under 200	100.0	52.8	47.2	100.0	76.7	23.3

satisfactory, and has been used in most of the work done thus far. Water leaves this chamber through a 1½-in. pipe discharging free to atmospheric pressure. A similar chamber having a 3-in. outlet opening was found to give slightly higher capacity, and was entirely satisfactory when operating with a vortex-type underflow; its use prevented the formation of a sausage-type underflow, however. The fact that either a 3-in. outlet or a vent in the top of the overflow chamber prevents the formation of a sausage-type underflow suggests that a sausage-type underflow can be obtained only when there is a slight vacuum at the top center of the overflow chamber.

Driessen's[2,3] reports illustrate an overflow chamber that is nearly twice the diameter of the conical section and encloses the

An attempt was made to eliminate the overflow chamber entirely by attaching a 2-in., U-shaped pipe directly to the orifice plate, but this arrangement appeared to create more back pressure than the chamber and was abandoned.

Comparison of 10 and 20° Conical Sections

All of the early work with the cyclone made use of a conical section like that illustrated in Fig 1, having a total enclosed angle of 20°. Later a conical section with a total enclosed angle of 10° was installed and found to possess several definite advantages. Table 5 presents the results of two cyclone tests made with 10 and 20° cones on a slurry of low solids content, with the cyclone operating at 15 psi feed-nozzle pressure. The 10° cone handled 34.1 gpm in comparison with 28.4 gpm for the 20° cone, an increase of 20 pct in capacity. The pump speed was the same for these two tests. In a number of other comparisons of the 10 and 20° cones, with the feed pump operating at the same speed, the advantage in capacity of the 10° cone has ranged from 20 to 30 pct. Thus, although the 10° cone is longer it offers less resistance to the flow of slurry and therefore provides higher capacity.

The figures in Table 5 showing the distributions of solids by size fractions demonstrate that the 10° cone also gave a substantially higher recovery in the underflow product of all sizes of solids. In fact, with the 10° cone there was virtually no loss of material coarser than 200-mesh in the overflow water. The recovery of material coarser than 200-mesh effected with the 20° cone would have been much higher, however, if the overflow opening had been smaller. With a slurry containing less than 3 pct solids, and with an underflow opening of ⅞ in., the 1¼ overflow opening is too large for the 20° cone but is correct for the 10° cone.

Table 6 gives the results of a similar comparison between the 10 and 20° cones, but with the cyclone treating a slurry of much higher solids content. In these two tests the 10° cone handled about 30 pct more slurry than the 20° cone, but part of this advantage can be attributed to the lower solids content of the feed—15.5 pct for the 10° cone in comparison with 21.1 pct for the 20° cone. The recovery of coarse sizes effected by the 20° cone was good but did not match the more efficient operation of the 10° unit.

TABLE 6—*Comparison of* 10 *and* 20° *Cone Sections, with Cyclone Operating on Slurry of High Solids Content*

	Test 55—20° Cone			Test 56—10° Cone		
	Feed	Over-flow	Under-flow	Feed	Over-flow	Under-flow
Cyclone openings, in.	1 1/16	1¼	1	1 1/16	1¼	1
Feed-nozzle pressure, psi	15			15		
Throughput, water, gpm	24.6			32.4		
Throughput, solids, lb per hr	3280.			3000.		
Solids, pct	21.1	1.9	58.3	15.5	1.4	58.3
Distribution of solids, pct	100.0	5.9	94.1	100.0	6.8	93.2
Screen analysis, pct						
35 to 48	26.9	0.2	28.6	22.0	0.0	23.6
48 to 65	30.7	0.5	32.6	27.5	0.1	29.5
65 to 100	16.5	1.1	17.5	17.7	0.1	19.0
100 to 150	9.0	1.7	9.5	10.1	0.2	10.8
150 to 200	6.7	4.5	6.8	8.6	0.5	9.2
Under 200	10.1	92.0	5.0	14.1	99.1	7.9
Distribution of solids by sizes, pct						
35 to 48	100.0	0.0	100.0	100.0	0.0	100.0
48 to 65	100.0	0.1	99.9	100.0	0.0	100.0
65 to 100	100.0	0.4	99.6	100.0	0.1	99.9
100 to 150	100.0	1.1	98.9	100.0	0.1	99.9
150 to 200	100.0	4.0	96.0	100.0	0.3	99.7
Under 200	100.0	53.6	46.4	100.0	47.8	52.2

The fact that the 10° cone provides more efficient recovery of solids, even with the greater throughput obtained, may be caused largely by the greater distance between the points at which the products are removed from the cyclone, or it may be caused by the flow characteristics of the longer, steeper cone—which, is not known.

Abrasive Wear of Cyclone

No information is available from Driessen's reports concerning the seriousness of abrasion in the cyclone thickener, and experience in the laboratory has been too limited to provide a reliable indication of cyclone life. After several months of intermittent use the first cyclone built shows no wear of the cylindrical section through which the feed tube enters nor of the upper portion of the conical section. The cylindrical section is made of mild steel and the conical section is of stainless steel. Wear is evident, however, in the bottom 2 or 3 in. of the conical section and in the interchangeable, stainless-steel, screwed plugs that are used to change the size of the underflow opening. The fact that intermittent laboratory use over a period of a few months has caused measurable abrasion in the bottom portion of the cyclone suggests that in a commercial unit it probably would be severe. A design providing a renewable liner for the bottom portion of the cyclone may be required.

Capacity of Cyclone

As the high capacity of the cyclone constitutes one of its chief advantages over other types of thickeners, a direct comparison of capacity in relation to area is of interest. With the 5-in. diam cyclone handling 50 gpm of slurry—a throughput that has been exceeded a number of times—its capacity is 367 gpm per sq ft of area.

In a recent publication on the flotation of coal slurries Davis[6] describes the performance of a common type of thickener treating a slurry with a solids content and size composition similar to slurries that have been used in the cyclone. This thickener was 60 ft in diam and handled 4000 gpm, or 1.4 gpm per sq ft of area. Thus the capacity of this thickener was 1/260 of that obtainable in the cyclone. Moreover, the efficiency of solids recovery in this thickener did not equal that obtainable in the cyclone. Parton and Rubert,[7] in another report on the flotation of coal, present performance data for a thickener operating as a hydroseparator; the slurry treated would be amenable to treatment in a cyclone. The hydroseparator, 75 ft in diam, handled 7000 gpm or 1.5 gpm per sq ft of area. This capacity was 1/230 of that obtainable in a cyclone making a similar separation at equal efficiency.

While comparison of a 5-in. laboratory cyclone with 60- or 75-ft conventional thickeners may not be entirely reliable, the high cyclone capacities obtained in this investigation, coupled with the fact that Driessen[2] reports a plant-size cyclone to have a capacity 400 times greater than conventional thickeners, leaves little doubt concerning the advantage of the cyclone thickener in terms of capacity.

This comparison between the cyclone and conventional thickeners should not be construed to imply, however, that the cyclone is superior in all other respects. The cyclone is, in reality, a classifier capable of making an efficient separation on coal slurry at a size of about 200- to 400-mesh. Normally, two-thirds or more of the solids finer than 200-mesh escape with the overflow water, and therefore the cyclone is not capable of producing a clear overflow water from a slurry containing a high proportion of very fine solids. Dorr-type thickeners, on the other hand, can be used to produce clear overflow water from slurries containing solids virtually all finer than 200-mesh.

OPERATION OF CYCLONE ON CENTRIFUGAL-DRIER EFFLUENT

The cyclone was installed temporarily in the central cleaning plant of the Northwestern Improvement Co., Roslyn, Wash., to permit observation of its operation under cleaning-plant conditions. The first slurry tried in the cyclone was the effluent of a centrifugal drier, which contains 30 to 35 pct solids as it leaves the drier but is diluted to 15 to 20 pct solids to facilitate pumping to the plant settling cone. The object of these tests was to see whether the cyclone could treat the drier effluent to produce an overflow water containing less solids than that present in the water currently wasted from the settling cone, and at the same time produce an underflow coal which, depending on its quality, could either be added directly to the washed coal or retreated on tables. A cyclone functioning in this manner would reduce the amount of solids in the water recirculating through the plant—an objective sought at most washeries.

TABLE 7—*Results of Cyclone Tests on Effluent of Centrifugal Drier*

	Test 46			Test 42			Test 43		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Cyclone openings, in.	$1\frac{7}{32}$	$1\frac{1}{16}$	$\frac{3}{4}$	$1\frac{1}{16}$	$1\frac{1}{16}$	$\frac{3}{4}$	$1\frac{1}{16}$	$1\frac{1}{16}$	$\frac{3}{4}$
Feed-nozzle pressure, psi	50			15			50		
Throughput, water, gpm	28.5			22.7			35.3		
Throughput, solids, lb per hr	3000			22000			3200		
Solids, pct	17.0	6.0	61.8	16.5	5.4	63.8	15.3	5.6	65.4
Distribution of solids, pct	100.0	28.3	71.7	100.0	26.4	73.6	100.0	30.8	69.2
Ash content of solids, pct	27.2	36.6	23.5						
Distribution of water, pct	100.0	90.9	9.1	100.0	91.7	8.3	100.0	93.4	6.6
Screen analysis, pct									
Over 48	18.1	0.3	25.2	22.6	0.0	30.7	15.2	0.1	22.0
48 to 100	25.3	1.5	34.7	26.4	0.2	35.8	23.3	0.5	33.4
100 to 200	17.7	0.8	24.3	15.3	3.2	19.7	18.2	4.3	24.4
Under 200	38.9	97.4	15.8	35.7	96.6	13.8	43.3	95.1	20.2
Distribution of solids by sizes, pct									
Over 48	100.0	0.4	99.6	100.0	0.0	100.0	100.0	0.2	99.8
48 to 100	100.0	1.7	98.3	100.0	0.2	99.8	100.0	0.6	99.4
100 to 200	100.0	1.3	98.7	100.0	5.5	94.5	100.0	7.3	92.6
Under 200	100.0	70.9	29.1	100.0	71.5	28.5	100.0	67.7	32.3

Table 7 shows the results of three tests made on the drier effluent with the cyclone operating under several different combinations of adjustments. Test 46 was made with a feed-nozzle opening of $1\frac{7}{32}$ in. and a pressure of 50 psi, while tests 42 and 43 were made with a feed-nozzle opening of $1\frac{1}{16}$ in. and pressures of 15 and 50 psi, respectively. The greatest capacity was obtained with the larger feed-nozzle opening and the higher pressure of test 43, in which the cyclone handled 35 gpm of water and 3200 lb of solids per hr. The quality of the overflow and underflow products obtained in these tests was substantially the same; the overflow water contained from 5.4 to 6.0 pct solids and the underflow ranged from 61.8 to 65.4 pct solids. Virtually all of the solids coarser than 100-mesh were recovered in the underflow product, as was 92.6 to 98.7 pct of the material of 100- to 200-mesh size. About 70 pct of the material finer than 200-mesh was lost in the overflow water, however. Whether the solids contained in the overflow-water averaging 96 pct finer than 200-mesh could be recovered by retreatment in another cyclone is a problem that remains unexplored.

The data for test 46 show that the solids in the cyclone feed contained 27.2 pct ash while those in the overflow and underflow products contained 36.6 and 23.5 pct ash, respectively. This cleaning effect obtained in the thickening operation is caused by the fine clay reporting largely to the overflow product. The extent of the cleaning effect obtained with a particular slurry depends upon the amount of clay present, for impurities of coarser particle size are more apt to enter the underflow than the overflow product.

The overflow water from the cyclone contained only 75 pct as much solid material as that carried by the overflow of the plant settling cone, and the solids were substantially finer—96 pct through 200-mesh for the cyclone and 83 pct through 200-mesh for the settling cone.

OPERATION OF CYCLONE ON SETTLING-CONE OVERFLOW

When the tests on the centrifugal-drier effluent were completed the cyclone was used to treat the overflow from a settling tank in the same washery. The settling tank is a 50-ft, 110,000-gal, conical tank of the type in general use at bituminous washeries to provide clarified water for recirculation. Most of the water overflowing the top of the cone is re-used in the plant, but a portion is bled off to prevent the solids from building up in the recirculated water. The overflow water that is wasted flows to a settling pond. The cyclone was tried on this waste slurry to determine whether some of the coarser solids could be recovered.

TABLE 8—*Results of Cyclone Tests on Settling-tank Overflow*

	Test 65—65 psi			Test 66—35 psi		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Cyclone openings, in	17/32	1¼	⅝	17/32	1¼	⅝
Throughput, water, gpm	51.2			38.1		
Throughput, solids, lb per hr	3080			2132		
Solids, pct	10.8	8.1	50.5	10.1	7.5	47.8
Distribution of solids, pct	100.0	70.7	29.3	100.0	70.2	29.8
Ash content of solids, pct		36.3	26.4		38.2	25.8
Distribution of water, pct	100.0	96.5	3.5	100.0	96.4	3.6
Screen analysis, pct						
Over 48	0.1	0.0	0.3	0.4	0.0	1.4
48 to 65	0.4	0.0	1.3	0.5	0.0	1.8
65 to 100	2.9	0.0	9.9	3.3	0.0	11.0
100 to 150	3.9	0.1	13.0	4.6	0.1	15.2
150 to 200	7.2	1.0	22.4	8.1	1.0	24.9
Under 200	85.5	98.9	53.1	83.1	98.9	45.7
Distribution of solids by sizes, pct						
Over 48	100.0	0.0	100.0	100.0	0.0	100.0
48 to 65	100.0	0.0	100.0	100.0	0.0	100.0
65 to 100	100.0	0.0	100.0	100.0	0.0	100.0
100 to 150	100.0	1.8	98.2	100.0	1.5	98.5
150 to 200	100.0	9.8	90.2	100.0	8.6	91.4
Under 200	100.0	81.8	18.2	100.0	83.6	16.4

Table 8 presents the results of two tests made at 35 and 65 psi feed-nozzle pressure. Except for the additional capacity obtained at the higher pressure, the results of the two tests are similar. The cyclone recovered about 30 pct of the solids present in the slurry, and produced an underflow product containing about 50 pct solids. The other 70 pct of the solids in the slurry—99 pct finer than 200-mesh—remained in the overflow water. The proportion of solids finer than 150-mesh that escaped in the overflow product was substantially higher than that experienced in treating coarser slurries.

A variety of feed-nozzle pressures and cyclone openings was tried but no combination of operating adjustments would yield a sausage-type underflow from this slurry. In treating a slurry containing solids that are over 80 pct finer than 200-mesh, the solids in the underflow product are too fine to maintain a bed in the bottom of the cyclone, and consequently the wetter, vortex-type of underflow is obtained.

OPERATION OF CYCLONE ON SPRINGBROOK SLURRY

The cylcone also was installed in the washery of the Springbrook Mining Co., Renton, Wash., to treat the overflow of a rectangular settling tank. This overflow product is a dilute slurry which currently is wasted to an outside settling pond. At many of the mines in this district the coal in such settling ponds is ultimately recovered by diverting the flow of slurry to a second or alternate pond while the coal in the first dries enough to permit loading it out, but the Springbrook slurry contains so much clay that it is an unsaleable, waste product.

Table 9 gives the results from one of a series of tests made on the slurry. The cyclone feed contained 4.0 pct of solid material, over half of which was finer than

200-mesh. From this slurry the cyclone yielded an underflow product containing 54.8 pct solids and an overflow water containing 1.8 pct solids. Solid material remaining in the overflow water was 99.8 pct finer than 200-mesh, which attests to the efficiency with which solids coarser than 200-mesh were recovered in the underflow product. No coal coarser than 150-mesh and only 0.6 pct of the 150- to 200-mesh solids present in the feed escaped with the overflow water. Material finer than 200-mesh was divided 76.1 pct to the overflow water and 23.9 pct to the underflow product.

TABLE 9—*Results of Cyclone Test on Springbrook Slurry*

	Test 59		
	Feed	Overflow	Underflow
Cyclone openings, in	1 1/16	1 1/4	13/16
Feed-nozzle pressure, psi	50		
Throughput, water, gpm	55.4		
Throughput, solids, lb per hr	1200		
Solids, pct	4.0	1.8	54.8
Distribution of solids, pct	100.0	41.9	58.1
Ash content of solids, pct	33.8	59.1	15.5
Distribution of water, pct	100.0	98.0	2.0
Screen analysis, pct			
48 to 65	5.6	0.0	9.6
65 to 100	12.6	0.0	21.6
100 to 150	13.0	0.0	22.4
150 to 200	13.9	0.2	23.8
Under 200	54.9	99.8	22.6
Distribution of solids by sizes, pct			
48 to 65	100.0	0.0	100.0
65 to 100	100.0	0.0	100.0
100 to 150	100.0	0.0	100.0
150 to 200	100.0	0.6	99.4
Under 200	100.0	76.1	23.9

The cleaning effect that accompanies the thickening of slurries containing considerable clay is especially evident in the results of this test. The clay is concentrated in the sizes finer than 200-mesh and since these sizes report predominantly to the overflow product the coal recovered in the underflow of the cyclone is much cleaner than the solids in the untreated slurry. The feed to the cyclone contained 33.8 pct ash, the overflow solids 59.1 pct, and the underflow coal 15.5 pct. Thus the solids in the overflow water were as high in ash content as the refuse produced in many cleaning operations.

Some difficulty in maintaining a sausage-type underflow from the cyclone was experienced in this installation because of fluctuations in the amount of solids in the feed. When the solids dropped much below 3 pct the size of the underflow opening had to be decreased to prevent the formation of a vortex. Not enough underflow coal was being produced to maintain a bed in the bottom of the cyclone. In the laboratory the cyclone has operated satisfactorily on feeds as dilute as 0.3 pct solids but the solids were substantially coarser than those in the Springbrook sludge. Apparently the lower limit of feed solids at which the cyclone will operate with a sausage-type underflow and not require adjustment in the size of the underflow opening is determined by the size composition as well as the concentration of the feed solids.

Difficulty in maintaining a sausage-type underflow with fluctuations in feed solids is encountered only when the concentration of feed solids is low and the solids are fine. With feed solids containing 10 to 20 pct of material finer than 200-mesh, the concentration of solids in the feed can vary from about 1 to over 25 pct and the cyclone will continue to operate satisfactorily without requiring any change in openings. With feed solids containing about 50 pct of material finer than 200-mesh, the cyclone will operate satisfactorily without adjustment in the range from 4 to 20 pct feed solids.

At the Springbrook mine the underflow of the cyclone could be added to the 3/16-in. to 0 washed coal where neither its ash nor moisture content would be objectionable. Thus the cyclone would salvage a saleable product from material now wasted.

SUMMARY

This report describes an investigation made by the Bureau of Mines of a cyclone thickener developed in the Netherlands.

The investigation has not been completed, but experience with the cyclone so far has demonstrated that its capacity is several hundred times that of conventional thickeners. Coal slurries can be thickened to a solids content of 60 to 65 pct. The cyclone effects virtually complete recovery of solids coarser than 200-mesh, but recovers only about one-third of the material finer than this size. Since recovery of solids finer than 200-mesh is incomplete, the cyclone is unable to produce a clear overflow water. The principal factors affecting the performance of the cyclone are the size of the openings through which the products are discharged, the slope of the cone, and the pressure at which the unit is operated. Since the cyclone is inexpensive to construct, has no moving parts, and is cheap to operate, it should find a number of applications, not only in the coal-washing field, but throughout the whole mineral industry.

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DISCUSSION

CYCLONE ELIMINATION OF CONTAMINATING SOLIDS FROM PROCESS WATER

Introduction

D. A. DAHLSTROM[*] and R. W. MAESER[†]—Cyclones or centrifugal collectors have been used for many years in the separation of solids down to approximately five microns in size from gases.[8,9,10] Their wide application to this problem is due in part to their high elimination efficiency in the correct size range. However, the main advantage lies in the very low capital, operating and maintenance cost caused by their simple construction and lack of moving parts.

In contrast, the cyclone has found only minor use in dealing with solids and liquids, probably because of its inability to produce an absolutely clear overflow stream. Driessen's paper in 1939, noting the utilization of the cyclone for the thickening of loess in a heavy medium process and followed by later reports on its application to fine coal washing, represents one of the first major installations in the coal-washing field.[11,12,13] Recently, Yancey and Geer have further increased the possibilities of the cyclone by demonstrating its practicability as a thickener of fine coal slurries in preliminary dewatering of coal.[14]

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[8] References are at the end of the discussion.

The object of this report is to indicate another application of the cyclone and to enlarge upon its fundamental theory.

Operators in the Illinois and Indiana area have discussed with the authors the problem of contaminating solids in process and recycle waters of coal washeries. These particles may be as coarse as 4-mesh, but the greatest percentage is found below 28-mesh. Their properties and ash content will depend upon the coal processed, the method of mining, and the nature of the banded impurities, overburden and underburden of the coal. To illustrate, the percentage of fine sizes in a mined product will be a function of the friability of the coal and the included extraneous material. Secondly, some methods of mining and transportation will increase this size range to a greater extent than others. Thus an appreciable quantity of fines are present when the coal is first immersed in water. Further fines are created during the cleaning by degradation of the coarser sizes. However, the greatest trouble is caused by fines resulting from silt and shale. By their physical nature, they tend to disperse as a suspension when placed in water even though originally present in coarse sizes. In addition, these resulting particles are usually finer than 200-mesh and offer a greater problem of extraction than the coarser coal. It is apparent that a large amount of the initial and created fines will pass through screens, filters, centrifugal driers, and other dewatering equipment resulting in a build up of contaminating solids in the recycled water. The problem is particularly acute in the case of strip mines and in localities where acid water conditions exist or water is in short supply. With the former, appreciable quantities of overburden and underburden are mined with the coal and thus increase the content of clay, silt, and sand. When acid water is encountered, the operator usually prefers a closed water system to reduce the amount of treatment necessary. This entails a high percentage of recycled water, allowing a large build up of fine solids. One strip miner in the Indiana area, with a clay overburden and severe acid water conditions, related that solid build up was so intense in his plant, the spray water sometimes would not pass through the screens unless a layer of coal was present.

One of the frequent complaints resulting from the above condition is caused by the coating of the washed coal by a very thin layer of silt or clay. The deposit not only yields a poor "customer appeal" color, but offers industrial combustion problems due to decrease in the rate of ignition. Another problem is increased operating difficulty and lowered efficiency of hydraulic washing equipment when using a water varying in solid content and size consistency. Finally, suspended solids in water streams increase maintenance costs and offer operating troubles such as plugging of lines and sprays.

Studies on the cyclone were initiated in July, 1947, at Northwestern University, as it appeared to offer a possible solution to the problem at a low capital and operating cost. If particles down to 10–15 microns can be eliminated, a large percentage of the original solids, including practically all the silt which causes a good share of the trouble encountered in the use of recycle water, will be removed. The cleaned water will be of a more consistent and optimum nature and together with its decreased effective viscosity may alleviate the problem of clay deposition on the washed coal. In performing the work, fundamental data were also taken in order that results could be used on any application of the cyclone. To simulate operating conditions, actual samples of wash water sediments from different points in various coal-cleaning plants were employed in the experimental work. Washery waste sediments from the slurry pond of the Truax-Traer Coal Co. mine at Fiatt, Ill. were used in the majority of runs, but results were checked with other materials. While studies are still continuing, sufficient results have been obtained to justify the publishing of a preliminary report.

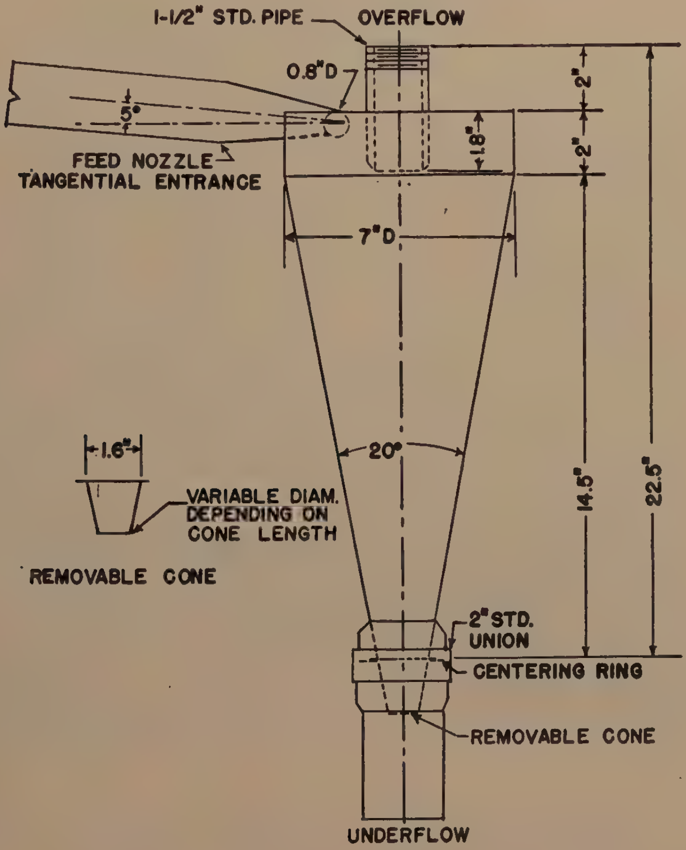

Fig 4—Construction of cyclone used in tests.

Description of Experimental Cyclone

The construction of the cyclone used in the tests is illustrated by Fig 4, and consists of a cylindrical section mounted above a truncated cone. The latter section has a maximum diam of 7 in., tapering to a diam of 1.6 in., with an included angle of 20°. The cylindrical ring is 2 in. high and 7 in. in diam, with a cover plate containing an overflow opening. The feed nozzle enters the cylindrical ring tangentially, as illustrated, and is a Venturi type nozzle with diam of 0.8 and 1.61 in. This nozzle is attached directly to the 1½-in. feed line. The entire conical section, cylindrical ring, cover and feed nozzle have been fabricated by rolling and brazing 0.04-in. sheet metal into the proper forms. The overflow discharge line is a standard 1½-in. pipe tapered at the fluid entrance in the cyclone. Two inches above the cyclone cover, a standard 1½-in. union is placed in the line. This union has been faced off to facilitate the introduction of orifices in the line at this point. The underflow discharge has been completed by attaching a 2-in. union on the bottom of the cyclone. A centering ring is placed in this union and nozzles with the same slope as the conical section of the cyclone are placed in this ring to control the discharge. These nozzles have been cut off at various distances, creating different discharge areas for the underflow stream. A 2-in. pipe is connected to the union to prevent spraying of the underflow.

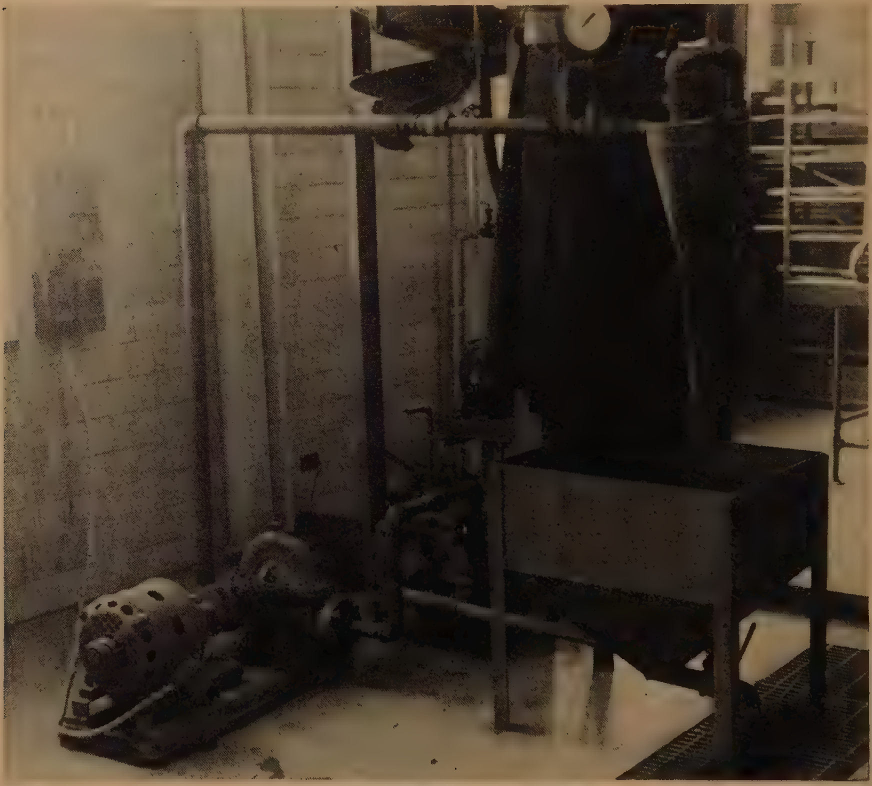

FIG 5—EXPERIMENTAL EQUIPMENT.

The slurry is pumped into the cyclone through the feed nozzle; the tangential velocity thus creates a spiral pattern of high centrifugal force. The solid particles of sufficient size and gravity are accordingly ejected outward to the walls and discharge in the underflow. The majority of the water, with uneliminated fine solids, moves radially inward and upward at an inner spiral located in the cyclone core to pass out the overflow.

Various volume distributions can be effected in the cyclone by either placing an orifice in the overflow discharge line or changing the nozzle in the underflow discharge line. Rates of throughput or centrifugal force acting in the cyclone were altered by using one or both of the slurry pumps in series or by placing an orifice in the feed line between the cyclone entrance and the pump discharge. A pressure gauge was installed in the feed line before the inlet nozzle to obtain line pressures. At this point it should be obvious that no moving parts except the slurry pump are involved and that construction is relatively simple.

Operating Procedure

As pictured in Fig 5, the experimental equipment consisted primarily of a collecting tank for the feed slurry, two slurry pumps, and the cyclone proper. The system is a closed circuit in which both overflow and underflow streams discharge into the collecting tank and are recirculated by the slurry pumps. The slurry pumps when used in connection with orifices can supply a capacity up to 60 gpm through the cyclone.

In performing the tests, the slurry is introduced into the collecting tank and circulated through the cyclone until equilibrium conditions are obtained. Time samples are collected simultaneously from both the overflow and underflow streams. Line pressure and character of the discharges are also noted. The samples are analyzed for solid concentration and particle size. The feed composition is calculated by combining the overflow and underflow streams.

Particle size above 200-mesh was determined by wet screening with U. S. Standard screens. The minus 200-mesh fraction was analyzed by the hydrometer method. This analysis determines the weight percent as a function of the equivalent diameter—the diameter of a sphere of the same specific gravity and volume settling at the same terminal velocity as the particle in question. Although this may not correspond to the screen diameter, unless the particles are spheres, it yields a size which depicts the nature of the particle under actual operating conditions. Thus it is a very convenient test for this application but must be carried out under standard conditions in order to give dependable results.[15] There are several testing variables encountered, and these can cause considerable error if sufficient care is not exercised. However, tests made in the same laboratory will give comparative results which are quite reliable and can be compared with other similar data. The experimental procedure consists essentially of screening out the coarse solids (over 200-mesh) and placing a suitable amount of the remaining slurry into a 1000-ml graduate. A special calibrated hydrometer is inserted and readings are made at various time intervals while the particles settle. From these readings and calculations using Stoke's law, it is possible to determine the equivalent particle size consist of the slurry.

Operating Factors

Preliminary investigations were confined to determining the effect of operating variables on elimination efficiency. The factors examined were type and concentration of the underflow stream, slurry concentration, throughput or centrifugal force, volume distribution between overflow and underflow streams, types of sediment and pressure drop through the cyclone. With this work completed, conclusions could then be applied to the construction and operation of full size equipment.

Type of Underflow Discharge—Initial experiments indicated that the underflow discharge pattern is extremely important in obtaining high elimination efficiency. As long as sufficient water is present in the underflow a vigorous vortex discharge with a high tangential velocity will be maintained. The critical water percentage will vary with the size consistency and gravity of the solids. With the sediments tested, approximately 50 pct by weight or 35 pct by volume of bone dry solids yielded the usual critical value. If the underflow outlet was restricted above this value, solids would issue in a very concentrated form but would retain none of the rotational energy of the cyclone. Appreciable quantities of the fine solids were then picked up by the overflow stream with an accompanying loss of elimination efficiency. This condition is typically referred to as an overloaded cyclone. On the other hand, the vortex discharge permitted only a minimum pick-up of fine solids by the overflow stream. The difference in the results between the two types of underflow discharges can be observed in Tables 10, 11, and 12. Tables 11 and 12 indicate typical results obtained with the vortex discharge while Table 10 strikingly illustrates the loss in efficiency with the overloaded discharge at similar operating conditions. The small saving in water to the underflow by the overloaded case is more than offset by the loss in solids, especially of the minus 100-mesh sizes, to the overflow stream. This same effect has been noted with gas cyclones wherein elimination efficiency was increased by prohibiting solids from building up within the unit.[8,16]

TABLE 10—*Effect of Cyclone Overloading on Elimination Efficiency*

	Run 11			Run 5			Run 7		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Inlet Centrifugal Force—Gravities	21			64			124		
Pressure Before Nozzle—psig	6			21			52		
Solids—Weight pct	13.3	2.3	56.3	11.2	5.7	60.3	9.6	1.6	62.0
Solid Rate—lb per hr	1,532	214	1,318	2,420	1,110	1,310	2,670	390	2,280
Solid Distribution—pct	100.0	14.0	86.0	100.0	45.9	54.1	100.0	14.8	85.2
Water Rate—gpm	20.0	18.0	2.0	38.3	37.5	0.8	50.5	47.8	2.7
Volume Distribution—pct	100.0	83.6	16.4	100.0	92.0	8.0	100.0	89.6	10.4
Solid Distribution by Sizes—pct									
+10 Mesh*	100.0	0.4	99.6	100.0	0.4	99.6	100.0	0.0	100.0
−10 + 20	100.0	0.6	99.4	100.0	0.1	99.9	100.0	0.2	99.8
−20 + 40	100.0	2.1	97.9	100.0	0.2	99.8	100.0	0.4	99.6
−40 + 60	100.0	5.3	94.7	100.0	0.8	99.2	100.0	1.2	98.8
−60 + 100	100.0	12.6	87.4	100.0	8.6	91.4	100.0	2.1	97.9
−100 + 140	100.0	24.5	75.5	100.0	26.4	73.6	100.0	4.8	95.2
−140 + 200	100.0	38.0	62.0	100.0	44.0	56.0	100.0	20.1	79.9
−200	100.0	76.0	24.0	100.0	85.0	15.0	100.0	66.0	34.0
Screen Analysis—pct									
+10 Mesh	5.8	0.2	6.7	1.8	0.1	3.2	3.0	0.0	3.5
−10 + 20	29.8	1.4	34.6	10.5	0.1	19.3	20.2	0.3	23.7
−20 + 40	27.3	4.1	31.1	13.8	0.1	25.5	25.9	0.7	30.4
−40 + 60	12.4	4.7	13.7	8.8	0.2	16.1	13.4	1.1	15.6
−60 + 100	6.7	5.9	6.8	7.6	1.4	12.8	10.1	1.4	11.6
−100 + 140	2.4	4.0	2.1	3.6	2.0	4.8	3.7	1.2	4.2
−140 + 200	1.8	4.8	1.3	3.5	3.3	3.6	3.4	4.7	3.2
−200	13.8	74.9	3.9	50.4	92.8	14.7	20.3	90.6	8.1

* U. S. Standard.

Yancey and Geer appear to have compromised the two conditions by obtaining a rotating "sausage" discharge from the underflow. By this method, elimination of 100-mesh to 200-mesh fractions was only slightly decreased, although the minus 200-mesh fraction was seriously affected. This type of discharge could not be obtained with the sediments tested probably because of the finer size consistency of the slurry and the larger diameter cyclone used. Yancey and Geer were primarily interested in dewatering, however, as opposed to this study where elimination efficiency is of primary importance.

Effect of Slurry Concentration—Theoretically, the solid concentration of the slurry should have no appreciable effect on the elimination obtained by centrifugal force. If sufficient force is present, a particle should be ejected to the underflow. However, as pointed out by Driessen, the particle must be eliminated from a suspension of higher gravity and viscosity than pure water and thus will act as though settling

TABLE 11—*Effect of Feed Slurry Concentration on Extraction Efficiency*

	Run 12			Run 13		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Inlet Centrifugal Force—Gravities	23			23		
Pressure Before Nozzle—psig	6			6		
Solids—Weight pct	10.7	2.0	51.5	15.5	2.8	45.0
Solids Rate—lb per hr	1,305	195	1,110	1,890	242	1,648
Solid Distribution—pct	100.0	15.0	85.0	100.0	12.7	87.3
Water Rate—gpm	20.8	18.8	2.0	20.5	16.5	4.0
Volume Distribution—pct	100.0	85.5	14.5	100.0	74.0	26.0
Solid Distribution by Sizes—pct						
+10 Mesh*	100.0	0.0	100.0	100.0	0.0	100.0
−10 + 20	100.0	0.0	100.0	100.0	0.0	100.0
−20 + 40	100.0	0.0	100.0	100.0	0.0	100.0
−40 + 60	100.0	0.1	99.9	100.0	0.2	99.8
−60 + 100	100.0	1.3	98.7	100.0	1.3	98.7
−100 + 140	100.0	2.5	97.5	100.0	2.8	97.2
−140 + 200	100.0	4.2	95.8	100.0	3.4	96.6
−200						
Screen Analysis of Streams—pct						
+10 Mesh	3.2	0.0	3.8	3.8	0.0	4.4
−10 + 20	16.5	0.0	19.4	24.0	0.0	27.5
−20 + 40	26.0	0.0	30.7	24.5	0.0	28.1
−40 + 60	14.4	0.1	16.9	12.8	0.2	14.6
−60 + 100	9.0	0.8	10.4	7.7	0.8	8.7
−100 + 140	3.7	0.6	4.2	3.1	0.7	3.4
−140 + 200	3.2	0.9	3.6	2.7	0.7	3.0
−200	24.0	97.6	11.0	21.4	97.6	10.3
90 pct Elimination Particle Size—Microns		32.0			28.0	
50 pct Elimination Particle Size—Microns		26.0			24.0	

* U. S. Standard.

TABLE 12—*Effect of Centrifugal Force and Volume Distribution on Coarse Size Extraction Efficiency*

	Run 16			Run 2			Run 10		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Inlet Centrifugal Force—Gravities	132			72.6			16.3		
Pressure Before Nozzle—psig	54			21			5		
Solids—pct By Weight	3.5	1.3	5.6	16.0	2.9	32.9	9.2	0.8	43.6
Solids Rate—lb per hr	965	182	783	3,501	367	3,140	1,018	70	948
Solids Distribution—pct	100.0	18.8	81.2	100.0	10.4	89.6	100.0	6.6	93.4
Water Rate—gpm	54.0	26.8	27.2	36.7	23.9	12.8	20.0	17.5	2.5
Water Distribution—pct	100.0	49.6	50.4	100.0	65.2	34.8	100.0	87.8	12.2
Volume Distribution—pct	100.0	49.1	50.9	100.0	59.7	40.3	100.0	83.2	16.8
Solid Distribution by Sizes—pct									
+10 Mesh*	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0
−10 + 20	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0
−20 + 40	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0
−40 + 60	100.0	0.2	99.8	100.0	0.0	100.0	100.0	0.2	99.8
−60 + 100	100.0	0.5	99.5	100.0	0.2	99.8	100.0	0.5	99.5
−100 + 140	100.0	1.1	98.9	100.0	0.7	99.3	100.0	1.6	98.4
−140 + 200	100.0	2.0	98.0	100.0	1.1	98.9	100.0	2.6	97.4
−200	100.0	32.8	67.2	100.0	33.4	66.6	100.0	27.0	73.0
Screen Analysis—pct									
+10 Mesh	0.1	0.0	0.1	5.0	0.0	5.6	2.7	0.0	2.9
−10 + 20	5.2	0.0	6.4	18.1	0.0	20.2	19.6	0.0	21.0
−20 + 40	14.0	0.0	17.2	19.5	0.0	21.7	25.7	0.0	27.4
−40 + 60	10.1	0.1	12.4	11.4	0.0	12.7	12.4	0.3	13.3
−60 + 100	6.9	0.2	8.4	7.7	0.2	8.6	8.7	0.7	9.3
−100 + 140	3.3	0.2	4.0	4.5	0.3	5.0	3.6	0.9	3.8
−140 + 200	3.6	0.4	4.4	3.1	0.3	3.5	3.0	1.2	3.1
−200	56.8	99.1	47.1	30.7	99.2	22.7	24.3	96.9	19.2

* U. S. Standard.

through a fluid of specific gravity of 1.0 to 1.3 depending on the nature of the feed slurry.[12]

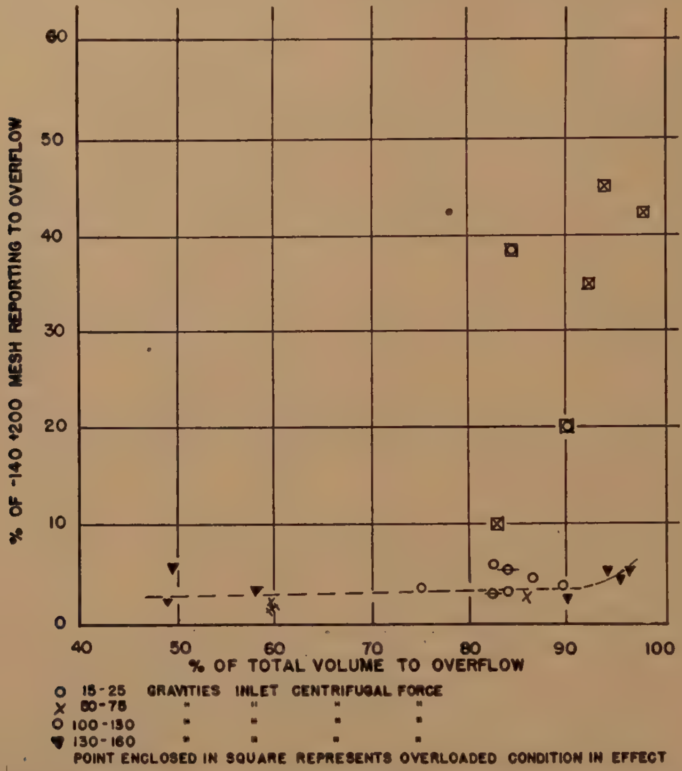

FIG 6—PER CENT OF 140-MESH TO 200-MESH PARTICLES VS. PERCENT OF TOTAL VOLUME REPORTING TO THE OVERFLOW.

Several runs were made at different slurry concentrations and throughputs, and in all cases, little, if any, effect of solid content could be observed. Table 11 is an illustration of two such runs and indicates that particle elimination within a size range remains constant. It was also observed that relatively wide fluctuation in feed slurry concentration did not have as pronounced an effect on the underflow concentration unless the cyclone became overloaded.

Centrifugal Force and Volume Distribution—Centrifugal force created in the cyclone is naturally one of the major factors in determining the elimination efficiency. Since this force is directly proportional to the square of the tangential velocity and inversely proportional to the radius of curvature, an increase in the volume throughput or inlet tangential velocity should influence centrifugal force. As discussed later, the controlling or critical centrifugal force does not occur in the peripheral ring of the cylindrical section, but at a radius well within the cyclone. Due to the nature of the flow pattern within the cyclone, the critical centrifugal force probably does not increase as the square of the volume throughput. Nevertheless, it does vary with the inlet velocity and accordingly a series of tests were performed at different throughputs. The inlet centrifugal force could easily be calculated for each run as the volume rate was divided by the feed nozzle cross-sectional area to obtain the inlet tangential velocity. Squaring this term and dividing by the cylindrical radius and the normal force of gravity gave the inlet centrifugal force in the unit of number of standard gravities. By varying throughput, tests were made at inlet centrifugal forces between 12 and 156 gravities.

In performing these tests, volume distribution between the overflow and underflow streams was also varied to observe whether a low solid weight per cent in the underflow stream would not increase the elimination efficiency of the cyclone.

The effect of volume distribution and centrifugal force on sizes coarser than 200-mesh was studied first. Only a small or negligible influence of these two variables could be noticed on this size range. Fig 6 is a plot of per cent of the 140-mesh to 200-mesh particles reporting to the overflow against the percent of total volume reporting to the overflow, with points of various inlet centrifugal force indicated. It is readily apparent that an almost horizontal line could be drawn through the indicated points with a vortex type discharge showing the relative independence of elimination efficiency with this size on these two factors. Table 12 illustrates this point. Runs made at high volume splits to the overflow and lower inlet centrifugal forces are compared with a run made at a high inlet centrifugal force and low volume split. The latter case should yield the best elimination efficiency if these two variables are important. However, elimination within a size range was practically the same above 200-mesh.

Evidently when sufficient critical centrifugal force is present to safely eliminate more than 90 pct of a particular size, higher centrifugal forces and underflow volumes will not appreciably increase ejection. It was further noted the plus 200-mesh in the overflow was a light gravity particle possessing a low ash content.

In studying the effect of centrifugal force and volume distribution on the minus 200-mesh particles, size analyses by the hydrometer method were performed on both underflow and overflow streams. From the sedata, particle sizes that were 90 and 50 pct rejected to the underflow could be determined. The 50 pct size is usually referred to as the equilibrium particle size and is a convenient measure of the elimination efficiency of a cyclone.

Table 13 is a summary of the two percentage particle sizes arranged in order of increasing inlet centrifugal force. Under the most efficient operation, 90 pct particle elimination size occurred at 20 microns or lower, while the equilibrium particle size was approximately 11 to 15 microns. This separation for the particular application is excellent, especially when one compares the operating ease and costs, extremely short retention time, small area, and simplicity with that of the conventional thickener.

It is readily apparent from Table 13 that increasing the inlet centrifugal force was accompanied by a lesser decrease in the equilibrium particle size. To explain this seemingly contradictory phenomenon, it will be necessary to discuss the fundamental flow pattern and theory of the cyclone.

TABLE 13—*Effect of Centrifugal Force and Volume Distribution on Particle Size Extraction*

Run No.	Centrifugal Force Gravities	Volume Distribution pct to Overflow	Particle Size Elimination Efficiency	
			90 Pct	50 Pct
			Microns	Microns
26	15	90	50	25
32	21	84	35	27
33	23	86	32	26
34	23	74	28	24
27	52	85	25	12
31	64	92	20	15
30	69	82	30	13
29	72	60	23	14
38	116	82	19	14
37	132	49	13	11
28	146	90	18	11
35	156	95	16	13
36	156	96	20	15

Experiments have proved that two concentric spirals exist in the cyclone. The outer spiral moves toward the apex of the cone while the inner spiral travels upward to the overflow. In addition, fluid is moving radially inward to the inner spiral. Thus, centrifugal force created in the inner spiral must be sufficient to impart a settling velocity greater than the rate of fluid movement into this spiral if a particular particle is to be eliminated.

As the critical centrifugal force developed in the inner spiral is a function of its tangential velocity as well as the radius, studies are to be made on the cyclone flow pattern to determine these variables. Very thorough research has been conducted on gas cyclones in this connection, and undoubtedly much of the theory can be applied to the use of liquids.

With gases, tangential velocity distribution has been defined as a function of the distance from the cyclone center line by the following expression:[16,18]

$$V_i = k/r_i^{n*} \qquad [1]$$

For a perfectly frictionless rotating fluid, angular momentum would be conserved and the exponent n would equal 1. However, frictional forces in real fluids will tend to reduce velocity differentials between adjacent layers. Thus, a highly viscous fluid could conceivably rotate at a constant angular velocity with an exponent n of -1. For gases, n has been found to vary from 0.5 to 0.7, with the former value generally accepted. No values are available for water, but they probably lie between 0.2 and 0.5.

The outer radius of the inner spiral again has not been measured for liquids but for gases it is known to be roughly equal to the radius of the overflow duct. However, maximum tangential velocity does occur at a point within the inner spiral.

With the above discussion in mind, the equation for Stoke's law may be used to define the movement of particles from 10 to 30 microns.[9,18]

$$u = \frac{(\rho s - \rho)gD^2}{18\mu} \qquad [2]$$

It is only necessary to substitute for the acceleration of gravity g, the centrifugal force V_i^2/r_i. Thus.

$$u = \frac{(\rho s - \rho)V_i^2D^2}{18\mu r_i} \qquad [3]$$

and for a particle to be eliminated, u must be greater than the radial influx of fluid to the inner spiral.

In actual practice there are several factors which become more pronounced in preventing lower size elimination as throughput or inlet centrifugal force is increased. With gases, it has been observed that as inlet velocity increases, the exponent n remains fairly constant. However, the radius within the spiral where tangential velocity is equal to inlet velocity steadily moves inward. Therefore, maximum V_i will approach a limiting value and thus prevent further increases in the inner critical centrifugal force. Undoubtedly this also occurs with liquids. Secondly, as throughput rises, the velocity of fluid moving into the inner spiral (or u) must increase. Consequently the phenomenon exhibited in Table 13 does not appear inconsistent.

As a final statement in this connection, it has been found that increasing cyclone diameter generally increases the equilibrium particle size with gases. The effect is not serious though, and it is believed that similar conditions will be experienced for liquids.

No definite conclusion can be made on the effect of volume distribution on size elimination with the present data. There seems to be a small tendency to decrease the particle size eliminated with increasing volume splits to the underflow. This can be largely explained by the greater amounts of water diverted to the underflow which will carry the very fine particles in suspension. On the positive side, it may be stated that low volume splits to the underflow with consequent high solid concentrations does not seriously injure particle size elimination as long as a vortex discharge is in effect.

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* Nomenclature indicated at the end of the discussion.

TABLE 14—*Effect of Clay and Silt on Elimination Efficiency*

	Run 24			Run 28		
	Clay Sediment Inclusions			Silt Sediment Inclusions		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Inlet Centrifugal Force—Gravities	144			146		
Pressure Before Nozzle—psig	52			52		
Solids—Weight pct	9.18	5.08	63.7	6.85	1.85	41.5
Solids Rate—lb per hr	2,760	1,416	1,344	2,070	490	4,580
Solid Distribution—pct	100.0	51.3	48.7	100.0	23.6	76.4
Water Rate—gpm	54.5	53.0	1.5	56.2	51.6	4.6
Volume Distribution—pct	100.0	94.6	5.4	100.0	89.7	10.3
Analysis of −200-Mesh Fraction						
Pct of Total Solids in Stream	67.5	99.0	34.4	40.6	99.1	22.7
Distribution by Streams—pct of Total −200 Mesh	100.0	75.3	24.7	100.0	57.5	42.5
50 pct Elimination Particle size—Microns		10.0			11.0	
Pct of Fraction Finer than the 50 pct Size	75.4	97.5	7.6	45.9	91.0	4.2

Effect of Variation in Sediments—As all previous runs had been made with silt bearing sediments, it was necessary to prove results on other materials. Accordingly, tests were performed on the overflow water from the fine coal settling cone of the Northern Illinois Coal Corporation at Wilmington, Ill. The minus 200-mesh fraction of these solids is largely clay and therefore, of a much finer nature. Results of two representative comparative tests are given in Table 14. It will be noted that equilibrium particle size for the two sediments is approximately the same. The poorer separation of the total minus 200-mesh fraction with the clay sediments can be attributed to the larger content in the feed of material finer than the equilibrium particle size.

All the minus 200-mesh fractions tested had approximately the same specific gravities of 2.3. Therefore no comment can be made on the effect of particle gravity other than that the equilibrium particle size will undoubtedly decrease as specific gravity increases. It is safe to conclude, however, that the nature of the particle other than gravity, unless it is unduly oily, will have no effect on the equivalent diameter of separation.

Pressure Drop in the Cyclone—Cyclone friction loss and not pressure drop is the measure of the static pressure and power a pump must develop in operating the cyclone. If the relatively small amount of slurry issuing from the underflow is neglected, Bernoulli's equation may be applied between the inlet to the cyclone and the overflow discharge.

$$F_c = \frac{\Delta P}{\rho} + \frac{V^2}{2g} - \frac{V_o^2}{2g} - F_x \qquad [4]$$

Thus it is apparent that the cyclone friction loss is overcome by the entering kinetic energy as well as the pressure head. Undoubtedly this is the reason for the wide discrepancies obtained by attempting to relate pressure drop alone as a constant multiple of the inlet velocity head.

Friction loss in the cyclone consists largely of loss of kinetic energy of rotation in the inner spiral as wall friction is probably of little importance. The kinetic energy loss will be a function of the flow pattern within the cyclone which in turn is influenced by the inlet velocity, cyclone proportions, and inlet and overflow dimensions. In studies on gas cyclones, a theoretical equation for cyclone friction loss in number of entrance velocity heads was derived, assuming the velocity distribution exponent n to be 0.5.[9]

$$F_c' = \frac{r_a}{r_o r_d} \qquad [5]$$

The r values were believed to be a function of the thickness and height of the inlet stream and the overflow diameter and tests were made to substantiate the assumption. An excellent empirical correlation was obtained and could be stated:

$$F_c' = K\frac{bh}{e^2} \qquad [6]$$

The value of K was found to be a function of cyclone proportions and for the general case equaled 16.[17]

Due to the difficulty of formulating a theoretical friction loss equation for liquids until r values are determined, a direct empirical correlation similar to the above equation was attempted. Initially, water alone was used in order to reduce the number of variables. From rate and line pressure data, it was possible to calculate by Bernoulli's theorem and appropriate friction loss equations the inlet cyclone pressure and velocity head, friction drop in the exit duct and outlet velocity head. Thus, the calculated frictional loss in the cyclone proper could be isolated by Eq 4 and expressed as the number of entrance velocity heads. The value of K was then easily obtained by dividing the friction loss by the ratio of the squares of the nozzle inlet and overflow diameters. Results given in Table 15 indicate a consistent K value and friction loss as a direct function of the square of the inlet velocity. With higher volume splits to the underflow, K values were somewhat larger. However, this was largely caused by the inability to assign a rotational kinetic energy value to the underflow stream.

To observe the effect of solids on the K value, a number of slurry runs was made with the results as shown in Table 15. A maximum increase of 15 pct with an average of 8.9 pct was experienced indicat-

TABLE 15—*Cyclone Friction Loss Data*

Run	Gpm		Pressure, Psig		Cyclone Inlet Velocity Head Feet of Fluid	Calculated Cyclone Friction Loss		b^2/e^2	$K = F_c e^2/c^2$
	Feed	Over Flow	Line	Calculated Cyclone		Feet of Fluid	No. Veloc. Heads		
Water Only									
51	64.9	64.0	51	39.3	26.6	114.1	4.30	0.247	17.4
52	56.9	56.4	40	31.1	20.4	90.8	4.40	0.247	17.8
53	47.9	47.2	30	23.7	14.5	68.2	4.70	0.247	19.0
54	40.1	39.7	21	16.6	10.2	47.3	4.65	0.247	18.8
55	39.4	39.2	17.8	13.5	9.85	40.8	4.14	0.247	16.8
Average									18.0
Slurries									
27	34.5	29.4	16	12.6	7.53	35.8	4.75	0.247	19.2
28	58.1	52.3	52	44.3	21.5	111.6	5.19	0.247	21.0
100	64.0	61.1	57	45.3	25.9	124.4	4.81	0.247	19.5
101	61.9	58.6	56	44.8	24.2	119.7	4.94	0.247	20.0
102	65.0	60.6	59	46.9	26.7	128.2	4.81	0.247	19.5
103	49.9	48.5	35	27.6	15.8	74.0	4.70	0.247	19.0
104	65.1	60.1	57	45.4	25.9	124.0	4.79	0.247	19.4
Average									19.6

ing that a relatively small correction factor for slurries will be necessary.

The above runs were made in one cyclone with fixed proportions and inlet and overflow diameters. Undoubtedly the final equation for friction loss will also be a function of cyclone slope and diameter with possibly different exponents for the inlet and overflow dimensions. Accordingly, a cyclone is now being constructed which will permit the complete analysis of this important operational factor.

The authors do not claim that the K value as determined in this report can be applied without error. However, it will at least give a reasonable approximation for cyclones of the same included angle until the exact equation can be determined. By this method, the following steps should be employed in calculating the line pressure necessary to operate a particular cyclone at any throughput:

1. Calculate the cyclone friction loss by the formula,

$$F_c' = K\frac{c^2}{e^2}$$

and convert to feet of fluid. A suggested value for K is 19.6.

2. Determine the friction losses of the inlet nozzle and exit ducts by appropriate equations.

3. The line pressure at the nozzle in feet of fluid will then be equal to the sum of items 1 and 2 plus the cyclone exit velocity and minus the line velocity head.

Cyclone Application

From the preceding discussion, several proposals can be advanced concerning the design and operation of the cyclone.

Of prime importance is the practicability of using the cyclone for elimination of solid particles of sufficient size from water as well as for the preliminary dewatering of fine coal. It has been proven that a high elimination efficiency is maintained down to particles of about 12 microns equivalent diam for silt and clay sediments with the test cyclone. According to Stoke's law, coal of 1.4 gravity compared with that of silt of 2.3 could be eliminated with the same efficiency down to approximately:

$$12 \times \frac{(2.3 - 1.0)}{(1.4 - 1.0)} = 21.6 \text{ microns}$$

Thus, if an industry, such as in coal beneficiation or mining, can tolerate particles of this size in the re-cycle water, the cyclone offers a very inexpensive solution to a troublesome problem. Capacity of the 7-in. cyclone used in these studies was about 40 gpm at 20 psi pressure. This corresponds to 150 gal per sq ft of area per min. which is about 150 times that of the conventional thickener operating at the same efficiency.

From experiments conducted in these studies, it was ascertained that an increase in throughput did not assert an accompanying increase in centrifugal force and elimination efficiency after an inlet velocity of approximately 20 fps was attained. This predicates the possibility of using larger inlet nozzles with a corresponding increase in throughput. A second advantage of such an installation would be the reduction in frictional loss of the cyclone at similar throughputs as this factor is proportional to the inlet velocity head. Consequently, it might be possible that the same elimination efficiency could be maintained at a lower operating cost. Probably the best method of determining this economic balance is to design the cyclone to take easily changed insert nozzles and overflow ducts as well as underflow nozzles. Thus, by simple initial tests the correct diameters for optimum operation can be ascertained. The system also lends itself to maintenance problems as these three points offer the greatest abrasion wear. It will be a simple matter to install the proper insert when necessary.

For the cleaning of process water, it is recommended the cyclone be operated with a high solid concentration of 40 to 50 pct by weight in the underflow which will safely furnish a vortex discharge. By this method, the greatest amount of clean water will be returned and the cyclone will possess almost maximum elimination efficiency at the operating pressure.

If operating costs are very critical, a gravity head of 20 to 30 ft available in many coal washing plants could be utilized to service a cyclone. In order to obtain sufficient capacity, the sizes of nozzle inlet diameter and overflow diameter would have to be increased over that used in these studies. Although efficiency will be affected due to the smaller inlet velocity head and larger diameter cyclone, the 50 pct equilibrium particle size should be well below 200-mesh.

Summary

The report is a summation of the work on the cyclone completed to date at Northwestern University. The study was concerned primarily with the efficiency of elimination of solids although much of the theory can be employed in other applications. Results indicate the cyclone offers a very inexpensive solution to the problem of sufficiently cleaning up wash water for re-use. Silt and clay particles with an average specific gravity of 2.3 could be eliminated down to an equilibrium particle size of 10–15 microns with a 7-in. cyclone. Larger industrial cyclones will exhibit a somewhat lesser efficiency.

The effects of slurry concentration, type of underflow discharge, volume distribution, throughput, and nature of sediments were observed and interpreted for operating conditions. The importance of frictional loss in the cyclone was advanced as the true measure of pump requirements and a temporary empirical equation originally applied to gases for estimating this value was derived from these studies. Work is now in progress to clearly define all factors affecting this important operating variable. Applications of the cyclone are discussed and the advantages of low cost, high capacity, ease of operation, and simplicity are emphasized.

Acknowledgments

The investigation is being conducted in the Chemical Engineering laboratories of Northwestern University. The authors wish to acknowledge the interest and suggestions of R. L. Sutherland, combustion engineer of the Truax-Traer Coal Co. and to the Truax-Traer Coal Co., Northern Illinois Coal Corporation, and the Mount Olive and Staunton Coal Co. for their submission of suitable samples for test purposes.

Nomenclature

Unless otherwise noted, any consistent system of units may be employed; the English system is given by way of example.

b = width of gas cyclone entrance, feet
c = cyclone inlet nozzle diameter, feet
D = diameter of particle, feet
e = cyclone overflow exit diameter, feet
F_c = cyclone friction loss, feet of fluid flowing
F_c' = cyclone friction loss, number of cyclone inlet velocity heads
F_x = other friction losses, feet of fluid flowing
g = acceleration due to gravity, 32.2 ft per sec^2
h = height of gas cyclone entrance, feet
k = constant of velocity distribution defined by $V_i = k/r_i^n$
ΔP = pressure drop, psi
r_a = radius at which the spiral velocity V_i is equal to the entrance velocity V, feet
r_d = inner radius of inner spiral, feet
r_i = distance from cyclone centerline, feet
r_o = distance from cyclone centerline to inner edge of outer spiral in cyclone, feet
u = settling velocity of particle, fps
V = average lineal velocity of fluid stream at cyclone entrance, fps

V_i = rotational velocity of fluid stream at a radius r_i, fps
V_o = cyclone exit velocity, fps
ρ = density of fluid, lb per ft^3
ρ_s = density of solid, lb per ft^3
μ = viscosity, lb per ft-sec

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H. F. Yancey and M. R. Geer (authors' reply)—The experimental results presented by Messrs. Dahlstrom and Maeser are gratifying to the authors because they confirm the general performance characteristics established for the cyclone. The only significant feature in which their results differ from our own is their inability to obtain a sausage-type underflow. This should be attributed to the design of the cyclone overflow rather than to the particle size of the slurry tested as they suggest. In fact, all of the slurries they tested are within the size range amenable to sausage-underflow operation. Early in our own work an attempt to eliminate the overflow chamber of the cyclone by attaching a U-shaped pipe directly to the overflow orifice—the design adopted by Dahlstrom and Maeser—proved unsatisfactory for sausage-underflow operation and therefore was abandoned; such a design is entirely suitable for vortex-underflow operation, however. The question of whether a sausage-type underflow should be used depends entirely upon the cyclone performance desired; the sausage-type underflow provides virtually complete recovery in the underflow product of all solids coarser than 200-mesh but eliminates 50 to 75 pct of the finer solids, with the overflow water, and at the same time provides the driest underflow obtainable. Vortex-underflow operation, on the other hand, provides maximum recovery of solids of all sizes but yields a substantially wetter product.

The term "elimination particle size" employed by Dahlstrom and Maeser provides a convenient, concise means of characterizing the particle size at which the cyclone effects a separation. However, these figures must be regarded as only approximate, for the difficulties involved in measuring size distribution in the subsieve range are much too great to warrant complete confidence in data obtained by merely observing the change in density of a settling suspension.

The Operation of a Froth Flotation Plant on Washery-water Solids

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(New York Meeting, March 1947)

A FLOTATION plant was placed in operation in 1945 at the Tamaqua colliery of the Lehigh Navigation Coal Co. Inc., to recover fine anthracite which is discharged from the cleaning plant with the washery water.‡ In addition to the recovery plant, facilities were provided for the complete clarification of the washery water, recirculation of this clarified water for breaker use, and disposal of the refuse solids in a waste impounding dam. Fig 1 shows a view of the settling tanks and the flotation plant.

Flow Diagram

The flow diagram for the complete recovery project is given in Fig 2. All the waste water from the cleaning plant is collected in a sump *A* outside the breaker structure, from which a 7000 gpm pump *B* delivers it through a 16 in. line to a 75-ft diam hydroseparator *C*. The hydroseparator, acting as a classifier, is set to give a separation at approximately 200-mesh. The minus 200-mesh overflow is discharged by gravity into the 180-ft thickener *D* for final settling of the solids. The classifier underflow sludge is pumped to the flotation plant by a 6-in. quadruplex diaphragm pump *E*. A 10-ft diam hydroseparator *F* is used to make a second classification on the sludge solids at approximately 28-mesh. The underflow from this classifier containing the coarse particles is elevated by a 6-in. diaphragm pump *G* and is discharged on two 4- by 6-ft vibrating screens *H*. These screens are dressed with approximately 28-mesh cloth. Screen oversize product flows directly to the centrifuge that dewaters the cleaned froth. The screen underflow and the overflow from the 10-ft classifier flow to the flotation circuit.

The flotation feed is conditioned in a 12 by 12-ft agitator tank *J* at a pulp density of 1:2.33 (30 pct solids). Reagents are fed to the circuit by reagent feeder *I*. The conditioned pulp is then diluted to a pulp density of approximately 1:5 (18 pct solids) and is fed to two banks of four 100-cu ft froth flotation cells *K* for primary coal recovery and cleaning. Additional reagents from reagent distributor *L* are added to the feed weir-box of Nos. 2, 3, and 4 cells of each bank of flotation machines. Clean coal removed from these cells flows directly to the clean-coal sump. Dilute tailings from the primary cells is rethickened in a 30-ft diam thickener *M* and is then retreated in a scavenger circuit for final recovery of coal. The scavenger circuit consists of a 6- by 6-ft agitator tank *N* and two banks of four 24-cu ft froth flotation cells *O*. Coal froth from these cells flows to the dewatering equipment and the tailings flows to a vertical refuse pump which delivers it to the 180-ft diam waste thickener *D*. Dewatering of the coal products is accomplished in a 54- by 72-in Bird centrifugal filter *P*. The cake product is dropped onto a conveyor and is loaded

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‡ A paper by the same authors on the pilot plant which preceded this plant appears in the *Transactions* AIME (1944) 157, 182–191.

directly into cars until the installation of a 11½- by 60-ft indirect-direct rotary heat drier *Q* can be made to further dewater this product. The effluent from the centrifuge and the overflow from the plant 30-ft thickener are pumped to the conditioner tanks for dilution water. The clarified water from the 180-ft diam waste thickener is delivered to the breaker for re-use by a 5000-gal pump *R*.

The thickened waste underflow solids are delivered to an impounding dam *S* by an 800-gpm sludge pump *T*.

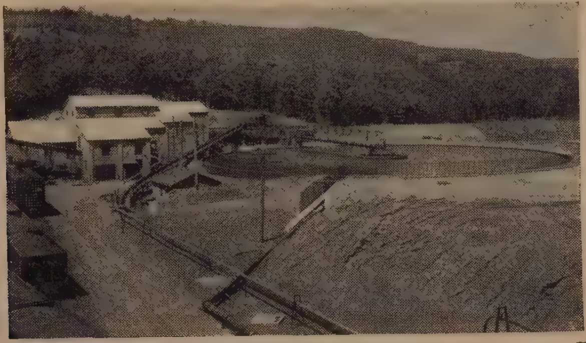

FIG 1—WASH-WATER CLARIFICATION AND FINE COAL RECOVERY PROJECT AT TAMAQUA COLLIERY OF THE LEHIGH NAVIGATION COAL COMPANY, INC.

PLANT OPERATION

Pumping and Hydroclassification

The waste water from the cleaning plant at the breaker is collected in a 10- by 12- by 9-ft deep sump. The top of the sump is covered with screen segments having 9⁄16-in. round-hole openings to screen out trash and oversize. A double-intake volute 7000-gpm pump equipped with machinable chrome wearing rings takes the water under a positive head from this sump and delivers it through a 16-in diam line to the 75-ft hydroseparator. Although a high velocity of 11.5 fps is carried in this line, considerable material moves along the bottom of the pipe and causes a large amount of wear. The line will be turned several times to obtain maximum line. Extra-heavy centrifugal cast pipe will be used to replace this pipe line when it is worn out in an attempt to obtain better service.

The 75-ft diam hydroseparator was constructed with a concrete bottom and wood stave-sides. The raking mechanism is suspended from the drive gear by two hangers, which protect the mechanism from severe overloads by raising the rakes when the torque on the mechanism is too great. In addition to this mechanism, a motorized rake-lifting device with push-button control in the plant was installed to iron out surges in the feed to the plant. The rakes can be rotated at ⅓, ⅔ or 1 rpm by use of different sheave combinations. Normal speed is ⅔ rpm or 157 fpm peripheral speed. Hydraulic water can also be used in the washing cone of the classifier to control the classification desired. During normal operation, cone wash water is not needed. Operating data for the hydroseparator are given in Table 1.

Recovery of the solids from the washery water and concurrent elimination of the finest particles of highest ash content with the overflow are accomplished in this classifier. The tank was designed to give sufficient upward current to overflow the colloidal clays with a minimum loss of plus 200-mesh coal. Data in Table 1

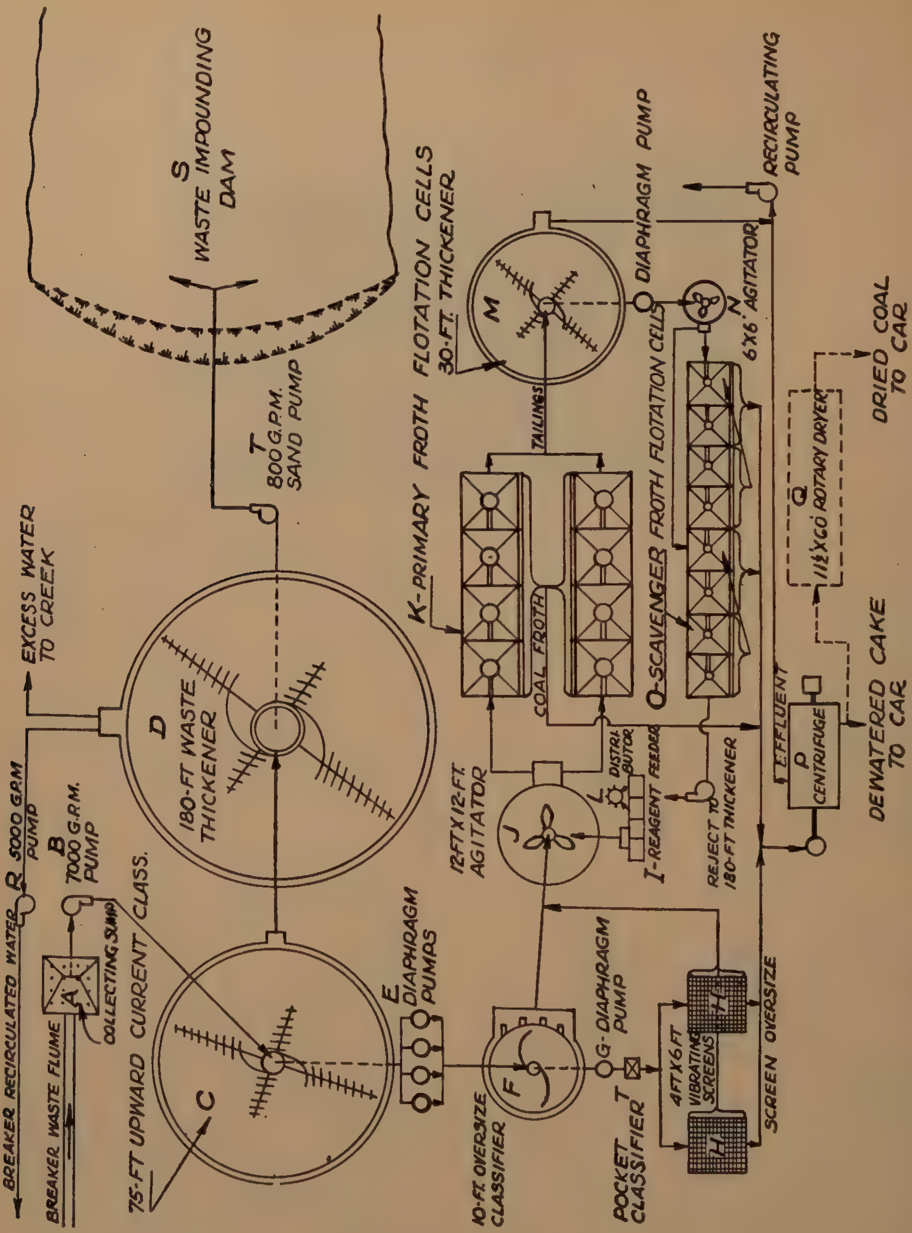

Fig 2—Flow diagram of flotation plant.

TABLE 1—*Classifier Operating Data*

Hydroseparator Operation (75 ft diam)	Hydro Feed	Underflow	Overflow
Gallons per minute	7,000	560	6,440
Solids, pct	5.3	35.0	1.9
Solids, tons per hour	91.0	60.0	31.0
Recovery, pct	100.0	67.0	33.0
Ash, pct	32.3	25.4	42.9
Screen Analysis, Pct Wt.			
+10 Mesh	0.2	0.2	0
+14 Mesh	0.2	0.3	0
+20 Mesh	2.4	2.0	0
+28 Mesh	4.6	5.0	0
+35 Mesh	7.6	10.0	0.2
+48 Mesh	9.4	14.5	0.2
+65 Mesh	9.6	15.0	0.2
+100 Mesh	10.8	19.5	0.2
+150 Mesh	8.6	14.5	0.4
+200 Mesh	7.0	9.0	1.8
+270 Mesh	2.6	4.0	2.0
−270 Mesh	37.0	6.0	95.0

indicate that 85 pct of the minus 200-mesh particles were eliminated in the classifier overflow while only a small percentage of plus 200-mesh coal is lost.

Oversize Classification

Experience in the flotation of anthracite sludges indicates that the presence of plus 35- or 28-mesh particles in sufficient percentages has a tendency to deaden the froth and retard the recovery of clean coal. The coarser particles are the last ones to float, and as a result, retention time in the flotation circuit must be prolonged to make satisfactory recoveries. In order to improve the general flotation conditions and recoveries, a 10-ft diam hydraulic classifier is used to make a density classification on the sludge underflow from the 75-ft hydroseparator. Operation of this classifier is controlled to maintain as high a percentage of solids as possible in the overflow, since this product flows directly to the 12-ft conditioner where too dilute pulp conditions are not desirable. The combination agitator and rake mechanism in the classifier is rotated at 4 rpm. Approximately 100 gpm of hydraulic water is used in the underflow washing cone.

The coarse underflow product is discharged through a 6-in. diaphragm pump to provide positive regulation of the underflow and to re-elevate this material so that it can be passed over two 4- by 6-ft vibrating screens. Much of the high-ash fine material that settles with the coarser particles is removed with the underflow of these screens. Ton-Cap and Ty-Rod stainless steel screens with openings of approximately 0.02-in. width are used with good results. Operating data showing the performance of the classifier and screens are given in Table 2. These data show that approximately three fourths of the sludge solids were removed with the overflow of the classifier, whereas one fourth of the solids carrying most of the coarse particles were discharged with the classifier underflow. Most efficient screening of this underflow product was not possible, since a limited amount of spray water was used to prevent overdilution of the underflow discharged into the conditioner tank ahead of the cells.

Cleaning this coarser fraction by tables or spiral separators would undoubtedly be an improvement over the screens. A cleaner oversize product would be produced and the high-ash fines would be eliminated with the cleaner refuse instead of being discharged into the flotation circuit. The removal of the pyrites accompanying these high-ash fine particles would be particularily desirable since these pyrites become oiled and tend to float with the coal. In order to reduce the ash content of the 10-ft classifier underflow (particularly in the coarser sizes) without the use of extra cleaning equipment, a somewhat unique classifier was designed and installed ahead of the two screens as shown in Fig 3. The classifier is of the surface current type, consisting of a feed plate *A*, upper sorting chamber *B*, and the lower or tester column *C*. Hydraulic

Table 2—*Operation of 10-Foot Classifier and Two 4 by 6-Foot Screens*

	Classifier Feed	Classifier Overflow	Classifier Underflow Screen Feed	Screen Oversize	Screen Undersize
Gallons per minute	560	540	140		
Solids, pct	34.8	28.6	32.0		
Solids, tons per hour	58.7	45.0	13.7	6.8	6.9
Recovery, pct	100.0	76.6	23.4	11.6	11.8
Ash, pct	29.7	27.3	34.3	24.0	43.6

Screen and Fractional Ash Analyses

	Classifier Feed		Classifier Overflow		Classifier Underflow Screen Feed		Screen Oversize		Screen Undersize	
	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct
Retained on										
+ 8 Mesh	0.0		0.0		0.1	12.6	0.6	15.7		
+ 10 Mesh	0.4	22.2	0.0		1.6		3.0			
+ 14 Mesh	0.4		0.0		2.2		4.0		0.1	18.9
+ 20 Mesh	3.5	14.1	1.0	10.2	10.2	16.44	20.4	17.0	0.2	
+ 28 Mesh	5.3		2.2		13.5		25.2		2.7	
+ 35 Mesh	9.5	20.3	7.6	13.5	16.7	23.7	21.1	25.7	13.4	38.6
+ 48 Mesh	13.4		12.7		17.8		12.6		20.4	
+ 65 Mesh	14.2	30.2	15.0	22.4	15.0	58.0	6.9	50.9	20.4	
+100 Mesh	18.8		20.4		12.7		3.9		21.0	50.6
+150 Mesh	13.8	34.5	15.4	34.1	5.7	57.5	1.2	50.2	11.1	
+200 Mesh	8.0		9.7		2.3		0.6		5.7	
−200 Mesh	12.7	45.6	16.0	46.7	2.2	51.8	0.5	46.2	5.0	
Tons per hr +28 Mesh	5.63		1.50		3.8		3.6		0.2	
Recovery of +28-Mesh Material, pct	100.0		28.3		71.6		68.0		3.6	

water is introduced at the bottom of the teeter column by diffusing it through a perforated screen. A stem-type valve is used to control the discharge of refuse or classified oversize from the bottom of the teeter column. The classifier was originally designed with a float mechanism to automatically control the position of the stem or plumb-bob valve. However, a simpler and very effective control was found by using the cam action of the diaphragm pump. A chain was attached from the cam through pulleys to the top of the valve stem so that the valve was raised and lowered 45 times per minute. The degree of opening of the valve on each stroke is controlled by the amount of slack in the chain. This type of classifier shows promise for use in preparing classified feed for concentrating tables. When used for removal of high-ash particles, a typical underflow sample is as shown in Table 3.

Table 3—*Typical Underflow Sample*

Volume, gpm	11
Solids, pct	40
Tons per hour	1.1
Ash, pct	58.41

Size Analysis, Mesh	Wt, Pct	Ash, Pct
+ 6	0.9	48.34
+ 8	1.3	
+ 10	4.2	
+ 14	7.6	
+ 20	23.4	66.65
+ 28	18.8	
+ 35	14.8	63.3
+ 48	12.5	
+ 65	8.4	74.0
+100	5.4	
+150	1.7	67.0
+200	0.6	
−200	0.4	57.5
	100.0	

In the operation of the classifier, primary stratification takes place according to size and gravity as the pulp flows over the feed sole or plate. The partially stratified particles enter the upper sorting chamber, which sloughs off the lightest and finest particles. Final separation on the particles that settle in the upper chamber is accomplished in the teeter column.

Flotation Circuit

After the oversize particles are removed by the classification circuit, the plant feed sludge is delivered to a 12-ft diam by 12-ft high agitator tank where part of the collector and frothing reagents are added. Approximately 13½ min. retention time is provided in this tank for thorough conditioning of the oils with the sludge. A 36-in. diam propeller is rotated at 185 rpm to produce the necessary agitation. A sand relief pipe in this tank was converted into an air-lift to make possible more positive removal of the particles that tended to settle and build up on the bottom of the tank.

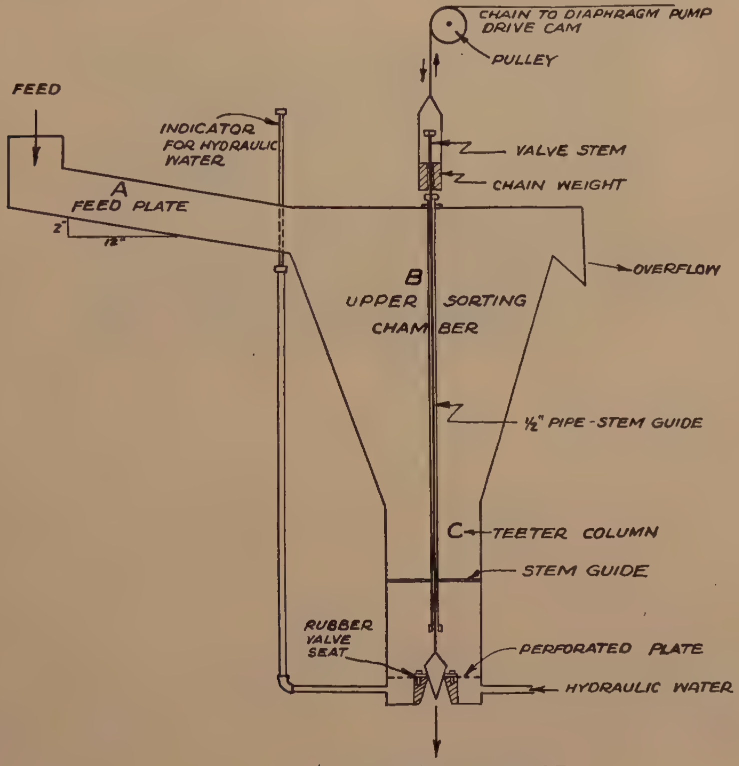

FIG 3—OVERSIZE CLASSIFIER.

The conditioned sludge is discharged from the agitator tank into a splitter box, where dilution water is added to reduce the proportion of solids in the pulp to approximately 20 pct by weight. The pulp is then divided into the two banks of No. 30 Denver Sub-A flotation machines. Each bank consists of four 100-cu ft capacity cells. Retention time of the pulp in these cells is approximately 6 min., during which time the bulk of the clean coal is recovered as a froth product. Four-bladed paddles are used to skim the froth from the tops of the cells. The cells are equipped with rubber-covered impellers, which are rotated at 250 rpm. Supercharged air at 6-oz pressure, produced by a 1000 cfm turboblower, is admitted to the standpipe of each cell to assist in the aeration of the pulp.

Operating data on the flotation circuits are given in Table 3. Since the bulk of the coal is recovered from these cells in the form of a froth carrying 40 pct solids by weight, the tailing from the cells is diluted to a low point, or 9.6 pct as indicated by the test data. In order to make as economical a retreatment of this pulp as possible for complete removal of the remaining coal, the primary tailing is rethickened in a 30-ft thickener tank and is then delivered to a scavenger circuit consisting of the flotation equipment originally used for pilot-plant purposes.

TABLE 4—*Operating Results for the Primary and Scavenger Froth Flotation Circuits*

	Primary Flotation Cell Feed	Primary Flotation Cell Coal	Primary Flotation Cell Refuse	Scavenger Flotation Cell Feed	Scavenger Flotation Cell Coal	Scavenger Flotation Cell Refuse	Composite Plant Product
Gallons per minute	1,000	280	720	360	25	335	
Solids, pct	19.0	40.6	9.6	15.4	41.5	11.7	
Solids, tons per hour	52.0	33.5	18.5	15.2	3.1	12.1	41.8
Recovery, pct	100.0	64.5	35.5	29.2	6.0	23.2	71.2
Ash, pct	30.13	13.04	60.82	61.35	24.05	70.72	14.5

Screen and Fractional Ash Analyses

Retained on	Mat., pct	Ash, pct	Mat., pct	Ash, pct	Mat., pct	Ash, pct	Mat., pct	Ash, pct	Mat., pct	Ash, pct	Mat., pct	Ash, pct	Mat., pct	Ash, pct
Mesh														
+ 10	0.0	11.27	0.0		0.0		0.0		0.0		0.0		0.3	14.26
+ 14	0.2		0.0		0.0		0.2	16.89	0.2	14.87	0.2	16.41	1.0	
+ 20	1.0		0.5	8.79	0.5	14.16	0.8		0.7		0.8		2.7	
+ 28	2.5		5.2		1.0		2.5		1.6		2.5		4.7	
+ 35	7.8	18.23	7.6	9.24	3.8	32.89	5.0	46.40	6.3	15.61	5.7	48.42	9.7	12.14
+ 48	12.9		13.8		7.1		9.7		11.5		8.5		12.3	
+ 65	15.0	27.89	14.0	12.83	10.3	60.52	13.4	66.08	12.9	24.59	12.2	73.69	14.0	14.75
+100	19.0		20.0		16.2		17.8		19.0		20.0		18.5	
+150	13.6	33.37	15.0	13.88	14.9	65.79	16.4	66.60	17.4	26.31	15.5	78.08	15.0	16.56
+200	9.6		9.7		9.9		11.7		12.5		11.2		9.0	
−200	18.4	46.45	14.2	19.50	36.3	70.04	22.5	72.07	17.9	30.55	23.4	79.40	12.8	21.77

The scavenger circuit is identical to the primary flotation circuit except that the equipment used is approximately one fourth as large. The rethickened primary cell reject is removed from the thickener tank with a 6-in. diaphragm pump. This pump discharges the thickener underflow into a 6- by 6-ft agitator tank, which is equipped with a 24-in. diam propeller rotated at 250 rpm. An additional 15-min. conditioning period is provided ahead of the scavenger flotation circuit. The flotation machine consists of eight 24-cu ft cells arranged into two parallel banks of four cells each, the same as the larger cells. The main difference in the operation of this flotation machine is that the froth from the last two cells of each bank is returned to the No. 1 cell for recleaning. Five minutes of flotation time is provided in these cells for final recovery of the coal that carries over from the primary circuit. Operating data in Table 3 show that the concentrate from the scavenger circuit is high in ash content. If market requirements indicate that a lower ash content for the composite plant product is desirable, this scavenger froth can be recleaned in separate cleaner cells or by combining it with the feed pulp being fed to the primary flotation circuit for recleaning in the large cells.

The flotation circuit used at the plant has the following advantages:

1. A small number of primary cells is needed, since a complete recovery of coal is not attempted in this circuit. However, the bulk of recoverable coal is obtained from the primary cells, since optimum pulp densities for most rapid flotation can be maintained.

2. Thickening of the primary dilute

tailings reduces the volume of sludge that has to be handled in the scavenger circuit. Small equipment can then be used for the final recovery step. As a result, the power requirement for the scavenger circuit is low. Furthermore, the thickener overflow can be recirculated for plant use.

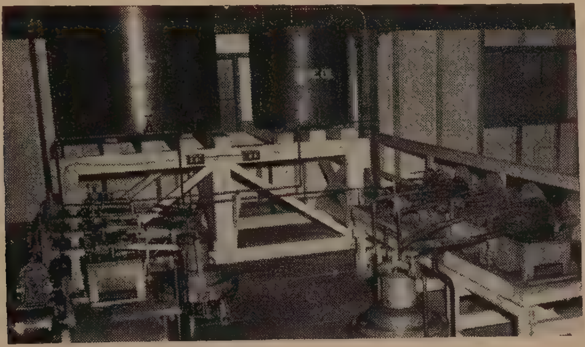

FIG 4—REAGENT FLOOR AT THE FLOTATION PLANT.

3. Efficient conditioning of the thickened solids in an agitator ahead of the scavenger circuit results in better flotation conditions and recoveries. Smaller amounts of reagents are accordingly required.

Reagent Feeding

Pine oil or an alcohol frother with a light fuel oil as the collector was found to be the most efficient reagent. Introduction of these oils into each circuit is accomplished by a four-compartment bucket-type reagent feeder. No. 1 frother and No. 2 collector compartments feed the reagents into the conditioning tank ahead of the cells, whereas No. 3 frother and No. 4 collector compartments deliver the oils into a reagent distributor, which in turn splits and delivers reagents to the individual flotation cells. By this arrangement, approximately one half of the total reagent is fed to the conditioner tank and the remaining half can be subdivided and fed to Nos. 2, 3 and 4 cells of each flotation machine. Duplicate feeding and distributing equipment is used for the primary and scavenger circuits. Fig 4 shows the arrangement of plant reagent head tanks, reagent feeders, and distributors. Table 5 shows the consumption of reagents in the two flotation circuits.

TABLE 5—*Consumption of Reagents*

Circuit	Oil	CC per Min	Lb per Ton Flotation Coal	Cost per Ton Flotation Coal
Primary......	Pine	144	0.47	$0.033
	Fuel	524	1.58	0.019
Scavenger....	Pine	22	0.07	0.005
	Fuel	112	0.34	0.004
Total.........	Pine	166	0.54	0.038
	Fuel	636	1.92	0.023
			Total cost reagent per ton flotation coal.......	$0.061

Product Dewatering

A 54- by 70-in. centrifugal filter of the solid bowl type has been used to reduce the moisture content of the coal froth produced by the flotation machines. Operating data in Table 6 show the performance of this machine.

These data indicate that a high recovery of the feed solids is accomplished in the filter. Solids lost with the effluent consist of high-ash slime particles. The effluent solids are approximately 97 pct minus

200-mesh in size and have an ash content of 45 pct.

The moisture content of the filter cake varies with the size consist of the feed solids. When the proportion of minus 200-mesh particles increases, the moisture content of the cake is higher. The amount of top-size particles present also definitely affects the dewatering results. When the flotation plant was first started, the filter was fed the froth concentrate from the cells only. Later, the plant oversize product was also fed to the filter, and improved dewatering performance was obtained.

TABLE 6—*Operating Data for 54- by 70-inch Centrifugal Filter*

Feed, tons per hour	41.0
Moisture, pct	57.2
Drycake, tons per hour	40.0
Moisture, pct	26.0
Recovery solids, pct	98.7
Effluent loss, tons per hr	1.0
Moisture, pct	97.8
Loss solids in effluent, pct	1.3

General data:
Conveyor, double spiral with 16 in. lead, slope 2° and 15°
Gear ratio, 20:1
Speed, 525 rpm
Feed-pipe length, 53½ in.
Pool depth, 3 in.
Power, 70 kw

The original single-spiral conveyor having a 10° uniform slope on the blade tips at the discharge end of the machine was replaced with a new design having a double spiral with two slopes on the conveyor tips. Dewatering performance of the machine with the new conveyor has been somewhat better, and it is hoped that the life of the stellited conveyor blades will be prolonged.

A 4- by 16-in. vibrating screen was recently installed in the plant for use in conjunction with the centrifugal filter. This screen was recently developed for dewatering coal sludges. The flat screen deck is covered with a special cloth made up of parallel stainless-steel rods. The first half of the screening surface is free of obstructions, but five dams are located between the screen panels on the last eight feet of the screen deck. These dams are sloped in the direction of the flow. The cloth was depressed with a predetermined radius on each side of these dams and the openings in the cloth were doubled in size in these depressions to permit more rapid draining of the water. The discharge end of the screen is elevated so that the material must work uphill, thus retarding the progress of the water and increasing the retention time of the coal on the deck. The dams also retard the progress of the coal across the deck and provide obstructions against which the coal backs up to produce a heavy bed of material. This thick mass of coal, backed up by the dams, results in squeezing of the water out of the layer of coal. The vibrating mechanism rotates at 1100 rpm, imparting a 5⁄16-in. stroke to the screen deck. The screen was equipped with cloth segments having ¼-mm openings for the first 4 ft and cloth with ½-mm openings for the remaining ½ ft of deck for the test results given in Table 7.

TABLE 7—*Operating Data for 4- by 16-feet Dewatering Screen*

	Screen Feed	Screen Cake	Screen Underflow
Solids, tons per hour	42	25	17
Moisture, pct	58.5	25.5	72.2
Recovery solids, pct	100.0	59.5	40.5
Ash, pct	14.1	13.5	14.4
Size analysis: retained on Mesh			
10	0.5	0.7	0.0
14	0.5	1.3	0.1
20	6.0	8.0	1.8
28	7.7	10.5	3.5
35	10.5	13.5	5.6
48	12.2	13.4	9.5
65	13.2	12.8	14.4
100	18.6	16.5	23.5
150	13.0	11.2	15.5
200	7.0	5.8	11.0
−200	10.8	6.3	15.1

These data show that considerable coal was lost through the screen with the water. On a product as fine as the flotation concentrate, these underflow losses cannot be eliminated entirely, since a reduction in cloth openings also results in a reduction of the dewatering capacity of the cloth. It appears evident that the screen would be incapable of operating alone to handle such a product as this one because of the underflow losses. However, this screen can be operated in conjunction with the centrifugal filter to make possible the maximum dewatering of the flotation product at a minimum cost. By feeding the screen all the flotation concentrate, approximately 60 pct is recovered as a dewatered screen cake. The screen underflow is then combined with the plant oversize product and pumped to the centrifugal filter. By this arrangement, the power consumed by the centrifuge is considerably reduced and the life of the spiral conveyor, the principal item of maintenance cost, is prolonged because of the reduced tonnage through the machine.

Another interesting possibility for using a dewatering screen of this type would be a circuit in which a thickener is used ahead of the screen. Operation of the screen on the thickened sludge should be considerably improved. Because of the reduced amount of underflow, screen underflow losses would also be reduced to a low amount and could be completely eliminated by returning the screen underflow to the thickener.

The dewatered product at 25 pct moisture forms a wet cake, which compacts into a solid mass when loaded into bins or railroad cars. As a result of the caking nature of the wet coal, unloading of the material from bins and cars requires extra labor and time. In addition to the handling difficulties, the high moisture content (reduced to approximately 18 pct by drainage) is generally objectionable to the consumers.

An extensive investigation was conducted to determine how this product could be dewatered with a minimum cost to a point where it would be satisfactory to the consumers. Prolonged storage to allow for natural drainage resulted in a final moisture as low as 13 pct. Very encouraging results were then obtained by blending this product with the next larger size, Anthrafine B, and allowing the mixture to drain. More rapid drainage resulted and the blended product could be handled without too much difficulty. The preparation of a large number of closely sized fractions of anthracite, which are sold at different rates by sizes (higher rates for larger sizes), prevents the blending of this product with large quantities of coarser coal. For Anthrafine B, the price is approximately the same as the flotation product, so that mixing of these two products is feasible.

The only method of making a sufficient moisture reduction on the product as it is produced appears to be by heat drying. Since the flotation plant at Tamaqua colliery had to be constructed at a site that does not allow for blending of the product with larger material and storage to permit drainage, heat drying equipment was ordered and will be erected at the plant to dry the product to a final moisture of 6 to 8 pct. In this moisture range, the fine coal no longer produces a cake and can be readily handled. A 11½- by 60-ft rotary drier of the indirect-direct type was ordered for this purpose.

Water Clarification and Impounding

The overflow from the 75-ft hydroseparator that carries approximately 31 tons per hr of slimes in 6500 to 7000 gpm of water and the flotation circuit reject sludge carrying approximately 15 tons per hr of refuse are combined and directed into a 180-ft diam thickener. This thickener produces an overflow carrying 0.01 pct solids or less, and an underflow product carrying 20 pct by weight solids. The overflow is pumped into the breaker as wash water for cleaning the larger sizes. The underflow is pumped to a waste-impounding dam for storage of these solids.

General Plant Operation

One of the characteristics of this plant is that it is dependent on the main breaker for its source of material. Fluctuations in the quantity and grade of material being fed to the main cleaning plant are reflected in a very short time in the quantity and nature of the material entering the flotation plant. Because of these variations, operation of the fine-coal recovery plant is somewhat complicated. Early in the operation of the plant it was found necessary to iron out the major surges in feed by installing a motorized rake-lifting device on the 75-ft hydroseparator. The plant operator can lift these rakes when surges of feed occur and thus store some of the material in the bottom of the hydroseparator tank.

The quantity of coarser particles in the feed sludge entering the flotation machines has a marked effect on the performance of the flotation circuit. When large quantities of coarse material enter the cells (in the range of 15 to 20 pct plus 28-mesh material) the froth becomes dead and very poor recoveries result. Best operation of the cells was obtained when essentially all the plus 28-mesh particles are removed from the feed ahead of the cells. A more desirable upper size limit would be 35- to 48-mesh. If the flotation feed could be limited to minus 35- or 48-mesh material, rapid flotation and high recovery efficiencies would result.

Power

Three-phase 25-cycle power is supplied at 11,000 volts to the plant transformers, which reduce this voltage to 440 volts for plant use. Most of the power consumed by the project is used by the pumps to deliver the slurry water to the plant, recirculate clarified water to the breaker and deliver the waste solids to an impounding dam. Total power consumed by the project is approximately 470 kw. Power requirements for various project purposes is given in Table 8.

TABLE 8—*Distribution of Power*

	Power Consumption, Pct of Total	Cost per Ton Product
Feed pump, 7500 gpm	17.2	$0.019
Breaker rec. pump 5,000 gpm	31.0	0.034
Waste-impounding pump, 800 gpm	11.8	0.013
Total pumps	60.0	$0.066
Classification	2.1	0.002
Conditioning	4.2	0.005
Flotation	13.6	0.015
Thickening	1.2	0.001
Product dewatering	11.4	0.013
Plant pumping	7.5	0.008
Total plant	40.0	0.044
Grand total	100.0	$0.110

Operating Costs

Accurate operating costs for the fine-coal project are not yet available because of the short time the project has been in operation. Operating costs in Table 9 were estimated on the basis of an average output of 525 tons per day. Labor costs were calculated for a 6-day work week using rates established by the agreement of June 7, 1946. Power and reagent consumption is based on actual figures for the time the plant has been in operation. The maintenance costs were calculated by using renewal costs for the equipment that required maintenance to date.

TABLE 9—*Operating Costs for Flotation Project*

	COST PER TON PRODUCT	
Flotation Plant:		
Labor	$0.130	
Power	0.044	
Reagents	0.061	
Maintenance, material and labor	0.072	
Total plant operation		$0.307
Pumping and impounding:		
Labor	0.048	
Power	0.066	
Maintenance, material and labor	0.030	
Total pumping and impounding		0.144
Total operating cost		$0.451

as well as an estimated cost of keeping all the equipment and facilities in operating condition.

These costs do not include the expense for the new heat drying equipment that is being installed at the present time. To these costs, the carrying charges to cover interest and amortization must also be added to get the complete operating costs for the project.

SUMMARY

The froth flotation project at Tamaqua colliery has been operating successfully to produce large quantities of fine coal by recovering and cleaning the material discharged with the waste wash water from the breaker. In addition to the recovery of coal, the wash water is clarified and returned to the breaker for re-use. The refuse solids from the recovery plant are impounded in a waste dam. With the operation of these facilities stream pollution with silt is completely eliminated at this colliery.

The froth flotation circuit used for cleaning the raw feed sludge has efficiently produced a satisfactory product at low cost. This system has been successful in handling feeds ranging from 40 to 60 tons per hour of solids and to produce 30 to 45 tons per hour of marketable coal. Daily production ranges from 500 to 550 tons per day. Direct operating cost for the flotation plant, only, is approximately $0.31 per ton of product.

The removal of coarser particles ahead of the flotation machines by using a hydroseparator in conjunction with vibrating screens has resulted in high recoveries and improved flotation conditions. Froth flotation of the sludge is accomplished with a minimum contact time and amount of reagents by using an intermediate thickening step, which enables the primary cell tailings to be rethickened and reconditioned before final recovery is made in scavenger cells.

Dewatering of the clean-coal product has presented the main problem in the production of this material, since the mechanical equipment used for this purpose is capable of reducing the moisture only to 25 pct. Heat drying appears to be the only means of further reducing the moisture content of the output as it is produced. Accordingly, heat drying equipment will be installed to produce a product with 6 to 8 pct moisture content. If it would have been possible to blend the flotation product with sufficient quantities of larger coal and to place it in storage to permit natural drainage, the heat drying step could possibly have been avoided.

ACKNOWLEDGMENTS

Thanks are expressed to the various members of the operating and engineering staffs of the Lehigh Navigation Coal Company Inc. for their assistance during the construction and operation of the recovery project. The authors are also indebted to the technical representatives of the various equipment manufacturers, whose suggestions aided materially in the development of the recovery plant.

DISCUSSION

(*J. Griffen presiding*)

S. A. FALCONER*—The authors of this interesting paper are to be complimented on their fine description of the operations of this pioneer commercial-size flotation installation in the anthracite region. Some novel features have been incorporated in the design and flowsheet of this plant; and the data on operation and costs reflect careful planning by the management, backed by pilot plant experience, which has insured successful results.

Upon reviewing this paper, several comments come into mind. It is evident from the experience of Lehigh Navigation Coal Co. as well as that of other companies with coal flotation plants, that careful preparation and control of flotation feed are necessary in order to secure high flotation efficiency. The presence of fines or slimes in the minus 200-mesh range are undesirable because they are reagent consumers, and because this material which is usually high ash is difficult to keep out of the finished flotation concentrate. On the other hand, coarse sizes, above 28-mesh, are more difficult to float. Higher quantities of reagents are necessary in order to float these coarse coal sizes and when sufficient amounts are added to insure their recovery, some sacrifice in grade results because of the tendency of the reagent to pull fine refuse along with the coal.

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* American Cyanamid Co., Stamford, Conn.

In addition to sizing the feed into the optimum range for flotation, it is even more important that it is fed at uniform rate and pulp density and properly conditioned with reagents before entering the flotation cells.

All of the foregoing conditions have been recognized and adequately dealt with in the Lehigh Navigation Coal Company's plant. The primary hydroseparator makes an efficient separation of the minus 200-mesh fines in the washery water, as shown in the data of Table 1. In addition, this apparatus is capable of compensating for the surges in volume of flow of washery waste which is one of the characteristics of operation of a breaker. For the removal of plus 28-mesh sizes, it is interesting to note that classification as well as screening is used. The secondary pocket classifier, which was developed to meet local conditions, is somewhat unique in its method of operation, and is a tribute to the ingenuity of the operating staff of this plant. It is interesting to note that the screens themselves are apparently not very efficient especially as regards removal of plus 48-mesh sizes, but as the authors point out, this is caused by the fact that only a limited amount of spray water can be used on the screens in order to prevent undue dilution of the underflow which discharges into primary rougher flotation cells.

In the flotation circuit itself, it is to be noted that only one large conditioner tank is used ahead of the primary rougher cells. The single conditioner at this point is apparently satisfactory for the Lehigh Navigation Coal Company's operations. However, it should be mentioned that some plants have found it desirable to use two or even three conditioner tanks in series, rather than one single tank of equivalent volume, in the interest of better conditioning reagent economy, and circumventing any possibility of short-circuiting of pulp during conditioning with reagents.

The flotation flowsheet is rather unique in that a thickening step is interposed between the primary and secondary rougher operations. It is to be noted that the rougher concentrates in this plant are not refloated in cleaner cells, although a somewhat similar effect is obtained in the case of the secondary rougher flotation circuit by counter-currenting the froths back to the head cells.

The use of the intermediate thickener in the Lehigh Navigation Coal Company's flotation circuit has permitted them to reduce the volume of pulp discharged from the primary rougher flotation cells and thus use flotation equipment which was at hand after operation of the original pilot plant. It would seem to be questionable whether this same type of flowsheet would be justified in the case of an entirely new plant where there would not be available the thickener and flotation machines which Lehigh Navigation Coal Co. happened to have. In considering the design of a new plant, the cost of the extra thickener and pumps would have to be balanced against the cost of a few extra flotation cells, and perhaps a slightly higher reagent consumption.

It is interesting to note that Lehigh Navigation Coal Company's flotation operations are conducted at about 18 to 20 pct solids. This seems to be an optimum pulp density for securing good grade of concentrates with higher recovery of coal in optimum time of flotation. In American Cyanamid Company's laboratory at Stamford, we have found that although there is sometimes an advantage, from the standpoint of recovery, in floating at higher percent solids, there is usually no advantage from the standpoint of flotation machine requirements or grade of finished concentrates. In other words, at higher percent solids the time of flotation is slightly increased, and there is more tendency to trap high ash fines.

In the Lehigh Navigation Coal Companys' flowsheet, the overflow from the intermediate thickener is returned to the circuit although just where is not clear from the flowsheet. It would be interesting to know just what effect this return flow has on the overall flotation results, inasmuch as we note from the data in Table 3 that there is evidently 6.3 tons per hr of solids of undescribed analysis in this overflow.

The data given in Table 3 also indicate that the concentrate obtained in the secondary roughing operations is low grade and has the effect of raising the ash content of the overall plant product. Of course, the actual weight recovery of this scavenger float product is rather low.

It is an interesting conjecture whether a classifier or hydroseparator rather than a thickener might not have more useful application between a primary and secondary rougher flotation circuit. In the case of some coal feeds treated in the American Cyanamid Company's laboratory, particularly those containing a high proportion of coarse material, we have found it advantageous to use a flowscheme involving conditioning of the original flotation feed with optimum quantities of reagents to remove the finer, more readily floatable coal, but not sufficient to float the coarser coal sizes. Following this initial float, the tailing is subjected to classification or screening to eliminate the finer high-ash sizes and then the coarser fraction is conditioned with more reagents and subjected to a second stage of flotation. In this second stage of flotation, the coarse coal is floated but there are substantially no fine high-ash sizes present to contaminate the concentrates, and, therefore, it is possible to make a high grade concentrate. On some types of coal this scheme of treatment gives higher overall recovery and cleaner concentrate than a straight roughing-cleaning procedure.

The methods used to feed and distribute the fuel oil and pine oil flotation reagents reflect careful planning and selection of equipment, which undoubtedly contribute to the low consumptions—only 0.54 lb of pine oil and 1.92 lb of fuel oil per ton of flotation concentrate.

One of the most interesting features of this paper is the discussion of the dewatering problem. Dewatering of fine anthracite coal has always been a problem and has undoubtedly held back the more general adoption of flotation in the anthracite region. The Lehigh Navigation Coal Co. investigators have contributed a great deal to the better understanding of this problem, and offer constructive suggestions as to the solution. Their experiments with apparatus such as a centrifuge and screens may eventually result in improvement in design of such equipment which will at some future time result in more efficient dewatering. However, for their own particular market requirements, they have been forced to adopt heat drying. Other companies faced with less exacting conditions may find thickening, followed by screening or filtration or perhaps simple drainage from stockpiled material will give adequate dewatering.

It is to be hoped that the successful outcome of Lehigh Navigation Coal Company's venture will result in further flotation installations by mining companies confronted with the dual problem of dealing with stream pollution and increasing the yield of fine coal for which the market seems to be increasing. Already, another large operator in the anthracite region has just recently placed in successful operation a 50 ton per hr flotation plant, and plans are under way for a similar size plant at one of its other breakers.

H. J. Gisler*—*Flotation*—The general consensus of opinion on coal flotation indicates that present day practice precludes successfully floating coal coarser than 28- or 35-mesh. The thoughts expressed in the Lehigh paper seem to confirm this assumption.

Flotation of coarse and fine coal together definitely is a problem. Conditions ideal for recovery of fine coal are not necessarily ideal for the coarser fractions, and the writer's experience with coal flotation in various plants as well as in the laboratory bears out this opinion. Dilute pulps and careful reagent control favor flotation of fine coal whereas coal coarser than 35-mesh is more readily floated at a higher density under more intense conditions.

Coarse coal has a tendency to form a heavy thick froth on the surface of the flotation cell whereas fine coal froth is more "alive" and free flowing. The former must be removed from the cell surface rapidly, and generally the usual froth paddles are inadequate to handle this character of material. Special means must therefore be provided in such cases.

A separate flotation circuit for handling the coarse fraction of coal may be worthy of investigation and it may be that this suggested arrangement may be more economical than the present screening system or the proposed gravity methods. The 10 ft Hydro underflow would make ideal feed for a coarse coal flotation circuit and, judging from experience, such a circuit would under ideal conditions recover a low ash coal, coarse as 8- or 10-mesh.

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* Denver Equipment Co., Denver, Colo.

Gravity Removal of Pyrite and High Ash Refuse—The hindered settling classifier described in the Lehigh paper is unique in its operation, particularly the method of opening and closing the reject discharge valve.

A pulsating mineral jig with a rotating water valve, properly synchronized with the action of the plunger diaphragm, would give a series of upward pulsations. This action is very effective in gravity separation. It may be more effective as a means for removing the coarse refuse and pyrite, both coarse and fine. Its water requirements would be considerably less than a straight hindered settling classifier and in the writer's opinion the action of such a jig or mineral selector would be superior.

H. F. YANCEY*—All three of the papers on the cleaning of coal by froth flotation are a welcome addition to the literature on this subject—primarily because they all present detailed operating information on actual plants.

In this field especially, even more than in experimental and development work fields, there is a dearth of information for the operator who seeks to determine the investment and operating costs required to recover clean coal from washery water. The managements of the companies at which the three plants are working, the authors, and the program committee of the Coal Division are to be congratulated upon making the information available.

All three papers deal broadly with the same subject, that is, the recovery of fuel values from the fine coal present in washery water. It is interesting to note that the plant at Tamaqua, working on anthracite slurry, is able to benefit the subsequent flotation operation by first removing the high-ash fines in a classifier of the hydro-thickener-separator type. As a result, the feed to the flotation plant contains only 18 pct of material finer than 200-mesh, whereas at Champion and Empire the character of the fines does not necessitate preclassification and the feed to the cells at these plants contains over 70 pct passing 200-mesh.

At both Tamaqua and Champion oversized material in the feed is undesirable. With the anthracite operation at Tamaqua this size is placed at 28- or 35-mesh, but with the bituminous operation at Champion, it is a little finer, namely, 48-mesh.

The total direct costs at the two operations giving such figures are not stated so as to be directly comparable, because different operating conditions and labor scales are involved, but at both plants the reagent cost is about the same, namely 4 to 7¢ a ton of coal at Champion and 6¢ at Tamaqua.

The use of a solid-bowl centrifuge at Tamaqua in comparison with the usual vacuum filter at Champion is an interesting development, but it is perhaps significant that part of the load at Tamaqua is being removed ahead of the centrifuge by a horizontal vibrating screen.

This is a new use for such apparatus and when observed on a recent visit to the plant showed a very thick bed of oversize on the screen, not to be compared in any way with the ordinary use of dewatering screens at bituminous washeries.

The high-speed elevators used in breaking down froth at Champion form an important development, as the handling of froth has been a major problem at the few washeries in the United States that have tried froth flotation. In these few plants the usual practice has been to add the froth to coarser washed coal in the clean-coal bin, thereby furnishing a natural filter bed for the froth, such as is practiced at Empire. This arrangement naturally results in lower fixed and operating costs.

In Europe, where the froth flotation of coking-coal sludge is common practice because such coal has a value of perhaps twice as much as coal in this country, other devices have been used for breaking down froth. One of these, known as a "foam destroyer," was made by Schüctermann and Kremer-Baum of Dortmund, Germany, a large coal-washery equipment manufacturer. By this method, the froth product is sucked into a vacuum tank so arranged that the material strikes a fixed plate located inside the tank near the intake, thereby releasing the air entrapped in the froth. The desired froth then flows by gravity to the ordinary drum of leaf-type filter. With this device vacuum and impact are combined as compared with impact alone at Champion.

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* U. S. Bureau of Mines, Seattle, Washington. Published by permission of the Director, Bureau of Mines, U. S. Dept. of the Interior.

J. A. Younkins, Jr.*—I should like to ask Mr. Gandrud if there is any recirculation from the overflow to the settling tank? Are the solid contents up around 16 or 17 pct in the overflow of the settling tank?

I would also like to know what is the ratio between the 400 gal feed to the flotation plant as compared to the total volume of settling tank overflow.

400 gpm to a 12-ft cell seems to be a rather high volume. Was the air-type machine selected because it had a higher capacity than the mechanical machine?

I should like to ask Mr. Parton if it is not true that the higher pulp densities which he described in his paper are caused more or less by the higher specific gravity of anthracite coal? Is not the actual volumetric density in the anthracite flotation cell comparable to that in the bituminous coal flotation cell?

Also, I should like to ask Mr. Parton whether he has had any experience in cleaning the sludge.

B. W. Gandrud—The water is recirculated but it is pumped up over the hill into a settling pond where it all starts out first, and then it goes down into the plant. Some make-up water had to be added to the settling pond. That was done by pumping clear water out of the river so that all the water going to the plant is clear water. It is a very unusual situation in that respect and that is the reason, I think, that the solids do not build up so much.

As to the total volume of slurry, about 500 gal are pumped up into the second pond on the other side of the hill and some slurry overflows into the sump which is picked up by the pump—just how much I do not know, offhand; it is about 100 or 200 gpm. That overflows and goes into another sump where the solids settle down again, so there is an opportunity for solids to settle out and they are settled out completely, and all this overflow of water always comes back.

J. A. Younkins, Jr.—Your project is really to recover solids that are being wasted in addition to the recovery that you are getting in your settling ponds?

B. W. Gandrud—That is right. Previously that material was being wasted in the settling ponds. It was never recovered but by this system we are recovering it.

As to the pneumatic machine, we just felt that it would have better possibilities with this diluted material with 2½ pct solids than the mechanical machine would have. It is a fairly cheap installation. The 12-cell costs $600 or $700, as I remember. It treats 400 gpm and we did not think we could do that with a mechanical cell.

J. Griffen—Mr. Parton, will you answer the questions of Mr. Younkins?

W. J. Parton (authors' reply)—In regard to the cleaning of our waste impounding dam, I must admit that little thought has been given to the possibility of removing material from this area.

The present dam is designed to have a 15 yr life. At the end of that time, we shall undoubtedly prepare another area for disposing of this waste and we will allow the material to remain in the dam as deposited.

J. A. Younkins, Jr.—Your pulp is consistently higher in density than is prevalent in bituminous practice. Is that caused by the differences in specific gravity of the two types of coal?

W. J. Parton—Yes, we would have fewer particles per gal of sludge as compared with a bituminous coal sludge at the same pulp density because of the higher specific gravity of anthracite. Our coal is approximately 1.65 sp gr.

J. A. Younkins, Jr.—Therefore, when you speak of concentrate anthracite at 30 pct solids, that would probably be equivalent to around 20 pct solids in bituminous practice?

W. J. Parton—I doubt if the difference in density caused by the difference in gravities is as pronounced as you suggest.

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* Duquesne Light Co., Pittsburgh, Pa.

Flotation Treatment of Washery Water at the Empire, Alabama, Mine of the DeBardeleben Coal Corporation

By B. W. Gandrud,* Member, and H. L. Riley,* Junior Member, AIME

(New York Meeting, March 1947)

A froth flotation unit was added to the DeBardeleben Coal Corporation's Empire washer at Empire, Alabama, in the fall of 1940 following an investigation of possible ways and means of recovering coal values from settling-tank overflow or slurry. Officials of the DeBardeleben corporation hoped that some practical and reasonably economical way could be found for recovering fine coal from overflow water, although there had as yet been very little progress along this line in the Alabama coal fields. The Bureau of Mines was asked to cooperate in the project through its Southern Experiment Station, at University, Alabama. This paper presents a brief description of the process and the installation that resulted from the investigation of the overflow water or slurry problem at the Empire mine. The experimental data included as a part of this paper were obtained during the course of the investigation in 1940. Since that time, however, the flotation unit has been running more or less continuously without significant operational changes and is still performing pretty much in accordance with the data obtained in 1940.

Flowsheet at Empire and Type of Coal Treated

The Empire mine is developed on the Black Creek bed near the northeast border of the Warrior field. The coal in this bed is extremely low-ash and although there is an almost complete absence of bone coal and partings, the bed is so thin that, with mechanized mining, a high percentage of rock from roof and floor inevitably gets into the run-of-mine coal. The thickness of the Black Creek bed throughout the part of the Warrior field in which it is mined extensively is rarely over 30 in. and probably would average about 25 in. The coal is of the high-volatile A rank and of excellent quality, but as a result of admixture of roof and floor material, the run-of-mine product usually contains 15 to 20 pct rock by weight. The following is a proximate analysis on an "as-received" basis of a typical seam sample from the Empire mine:[1] moisture, 3.0 pct; volatile, 34.6 fixed carbon 58.3; ash, 4.1

This sample showed 15,070 Btu per pound and 0.9 pct sulphur on a moisture-and-ash-free basis.

In the Empire preparation plant the lump sizes above 3 in. are removed and hand-picked on a Marcus conveying screen, the 3 to ⅜-in. size is removed by vibrating screens and washed in two Montgomery single-cell jigs, and the ⅜-in. to 0 slack is cleaned on four diagonal

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* Supervising Engineer and Assistant Coal Preparation Engineer, respectively, Coal Preparation Section, Southern Experiment Station, Bureau of Mines, University, Alabama.

[1] References are at the end of the paper.

deck tables. Under normal operating conditions of two shifts per day, the jigs and tables give a daily output of about 1000 tons of 3-in. to 0 washed coal. At the time of the slurry investigation the minus 1½-in. size in the washed jig coal was being removed by means of a trommel screen and run into the washed-slack bin with the minus ⅜-in. washed coal from the tables. The resulting 1½-in. to 0 slack was sold partly to steam plants and partly to by-product-coke plants.

Fig 1—View of Empire preparation plant with small flotation unit in the background to the right.

There are two settling tanks, one for the jigs and one for the tables. The overflow water from both tanks and from the elevator boot, into which both the jig and table refuse discharge, empties into a small surge tank from which it is picked up by a 500-gpm centrifugal pump. Through a 6-in. pipe line this pump boosts the slurry to the top of a high ridge back of the washer, where it is discharged and flows into a settling pond. This settling pond acts as a gravity reservoir from which the water supply for the entire preparation plant is obtained. The water in the pond is replenished as needed by means of a pump line from the Mulberry fork of the Warrior River. The 6-in. slurry line is tapped at a point slightly below where it crosses the top of the ridge, and about 400 gpm of slurry water is piped along the side hill about 100 ft to where it discharges by gravity into the flotation machine. Fig 1 is a view of the main preparation plant, showing also the small structure on the side hill which houses the flotation unit.

Experimental Work

At the start of this investigation it was recognized that there should be no difficulty technologically in recovering clean coal from the slurry water by means of froth flotation. The most difficult problem seemed to be how to adapt froth flotation to the small-scale requirements at Empire so that it would be practical and feasible from the dollars-and-cents standpoint. It appeared that the usual scheme of dewatering a flotation product by thickening and vacuum filtering would be too expensive. The only promising alternative seemed to be to use a bed of washed slack as a means of recovering and filtering the fines out of the flotation froth. The filtering action of slack is obvious around most wet preparation plants where practically clear water can be seen running or dripping out of the bottom of washed-coal bins and loaded railroad cars. It was, therefore, decided to try the simple expedient of running the flotation froth into the washed-slack bin while maintaining in the bin at all times a sufficient bed of the washed slack to serve as a filtering medium.

Samples of the slurry water were first obtained for tests with a small, 250-gram, laboratory, mechanical flotation cell. The percentage of solids in the samples varied from 1.7 pct to 3.2 pct, with an average of 2.6 pct. Ash analyses of the solids ranged from 12.4 to 20.5 pct, with 16.3 pct as an average. Screening tests showed the slurry solids to be extremely fine-sized. A typical screen-size test is shown in Table 1.

Table 1—*Screen-sizing Test on Solids from Settling-tank Overflow*

Size, Mesh	Wt, Pct	Cumulative Wt, Pct
Plus 35	0.3	0.3
35–48	0.9	1.2
48–65	2.0	3.2
65–100	3.4	6.6
100–150	7.9	14.5
150–200	7.5	22.0
Minus 200	78.0	100.0

It will be noted that 78 pct of the material is minus 200-mesh. This minus 200-mesh material analyzed 25.2 pct ash. The sizes coarser than 200-mesh gave a composite ash analysis of 3.8 pct.

All samples responded readily to treatment by froth flotation, using pine oil as the reagent. The results of a typical test with a 250-gram mechanical cell were as follows:

Product	Wt, Pct	Ash, Moisture-free Basis, Pct
Composite feed......	100.0	16.8
Cleaned coal........	76.4	3.3
Refuse.............	23.6	60.3

In this test pine oil was used as the reagent, at the rate of 0.5 lb per ton of solids in the feed.

At the conclusion of the laboratory tests an experimental 5-ft Forrester-type pneumatic flotation cell was built out of 2-in. planks at the mine and set up adjacent to the main preparation plant. Air for this machine was supplied by a size 21031 Sturtevant centrifugal blower of a type used for experimental purposes at the Bureau of Mines Southern Experiment Station. A cross section of the 5-ft cell is shown in Fig 2.

TESTS WITH EXPERIMENTAL UNIT

Many tests were run with the 5-ft machine to determine the effects of various factors such as rate of feed, volume of air and amount of reagent used. The results, although not quite so good as with the small laboratory cell, were nevertheless quite satisfactory on feed rates up to 125 gpm slurry. The amount of reagent used varied from 0.56 to 1.35 lb of pine oil per ton of solids in the slurry, and there seemed to be no particular advantage in using more than one pound per ton. Effects of varying the amount of air were reflected in the ash analyses of the flotation products and in the recoveries. With other variables held constant, an increase in the air volume always lowered the grade of the floated coal, giving a product of higher ash analysis, but on the other hand increasing the recovery by giving a refuse with less coal in it. Decreasing the air volume always resulted in exactly the opposite effect. Results of a typical test with the 5-ft machine were as follows:

Product	Wt, Pct	Ash, Moisture-free Basis, Pct
Composite feed......	100.0	15.4
Cleaned coal........	71.0	3.9
Refuse.............	29.0	43.6

The feed rate in this test was 94 gpm slurry. The percentage of solids in the feed was not determined for this run but was probably close to the average 2.6 pct. The pine-oil consumption was 0.73 lb per ton of solids in the feed.

After starting operations with the 5-ft machine, it soon became evident that the main problem would be the handling of the froth from the machine to the washed-coal bin. The machine was on the opposite side of the main preparation plant from the washed-coal bin, so that there was a distance of about 100 ft from the machine to the bin with practically no slope. An attempt to pump the froth over to the bin with a centrifugal pump was only partly successful. To facilitate pumping, the froth was run into a circular steel surge tank of about 250 gal capacity. The pump intake pipe was connected near the bottom of the tank, with the idea that by pumping at the proper rate the froth would be retained in the surge tank long enough to break up and be sufficiently liquid at the pump intake for the pump to handle it. However, at times the pump removed the product in the surge tank too rapidly and thus lowered the level of the liquid in the tank to approximately the level of the pump intake. Then the pumping would stop as a result of the froth's getting into the pump and the pipes. Following such a sequence of events it was sometimes necessary to employ a jet of fresh water inside the tank to get pumping started again. In spite of this difficulty, the 5-ft machine was operated long enough to show quite conclusively that the fine coal in the flotation froth could be recovered and dewatered in the washed-coal bin and long enough to suggest the next step in the investigation. Under average conditions the froth from the 5-ft machine represented a water volume of 8 to 10 gpm and its discharge into the slack bin had no effect on the appearance of the water running out of the bottom of the bin provided there was as much as a 2-ft or 3-ft bed of slack in the bin to act as a filtering medium. Under these conditions the drainage out of the bottom of the bin looked clear as tap water. The slack bin at Empire is built out of roughly matched 2-in. planks and holds about 60 tons of coal when full. The unloading gates in the bottom are flat, horizontal, steel gates which slide in and out of wooden frames.

As to the problem of handling the float product from the machine to the bin, it appeared that the only economical, reliable, and trouble-free system would be to have the flotation machine at a high enough elevation so that the froth would run by gravity into the washed-coal bin. A few experiments with this material showed that it would flow freely down a launder having a 13.0 pct slope. Consequently, when the company decided to purchase a 12-ft, factory-made, pneumatic or "air-lift" machine from the Southwestern Engineering Co., plans were made to install it high enough up the hill to the rear of the preparation plant to give a minimum 13 pct slope down to the top of the washed-coal bin. Along with this machine was purchased a Roots Connersville positive volume blower for supplying air. Air requirements for the 12-ft machine were expected to be about 600 cfm. As a part of the installation, an open wooden launder was built for fluming the float product into the washed-coal bin by gravity. It is noticeable in Fig 1 that the lower end of this launder has rotted and been replaced by a 5-in. cast-iron pipe. The 6-in. slurry pipe was tapped near the top of the hill, and the slurry feed flows by gravity through a pipe from there down to the flotation machine. Fig 3 shows a cross-section and longitudinal view of a Southwestern air-lift machine like the one installed at Empire.

RESULTS WITH THE 12-FOOT PNEUMATIC MACHINE

The operating results with the 12-ft Southwestern Engineering Co. machine do not differ significantly from those with the 5-ft experimental machine with the exception of its increased capacity. It has 24 linear ft of froth overflow lip compared with 10 ft for the 5-ft machine. The capacity of the 12-ft machine is somewhat higher than this relation indicates, being on the order of 400 gpm slurry. Figures are not available for comparing the two machines on a volume basis. An average set of samples from operations with the 12-ft machine on a feed of 410 gpm slurry showed the following recovery and ash analyses:

Product	Wt, Pct	Ash, Moisture-free Basis, Pct
Composite feed......	100.0	16.3
Cleaned coal........	74.2	4.4
Refuse.............	25.8	40.5

The froth product in this run contained 18.5 pct solids. Pine oil was consumed at the rate of 1.36 lb per ton of solids in the feed. On a normal feed of 400 gpm slurry, containing 2.6 pct solids, the recovery of cleaned coal is about 3900 lb per hr on a dry-coal basis. The amount of water discharged along with this amount of coal is about 35 gpm. The froth flows by gravity down the launder into the bin without any settling-out tendency or trouble of any sort, and the bed of washed coal in the bin apparently recovers and dewaters the solids effectively. Efficient recovery should not be expected, of course, unless a bed of coal several feet thick is maintained in the bin at all times. Under these conditions bin drainage does not seem to be impaired and water running out of cracks in the bottom of the bin looks perfectly clear. A check on the amount of solids in this drainage water has been made by catching a sample in a bucket suspended under the bin. The water showed a "solids" content of only 0.1 pct.

SULPHUR AND FUSAIN

There has been no real sulphur problem at Empire since the coal is of the low-sulphur variety. Seam samples show on an average about 1.0 pct total sulphur. However, two samples, one of the slurry feed and the other of the float coal, were analyzed for sulphur. The analyses showed 0.7 pct and 0.6 pct sulphur in the feed and float coal, respectively. There was apparently no tendency for the sulphur to concentrate in the cleaned coal.

One peculiarity about the coal in the Empire slurry water is its high fusain content. This is not surprising in view of the petrographic character of the Black Creek bed at Empire. Petrographic examinations by the Bureau of Mines on a column sample in 1932 showed that approximately 10 pct of the coal substance in the column was fusain (p. 15 of ref. 1). The fusain is distributed throughout the bed in lenses and partings of variable thickness, but generally not more than a few hundredths of an inch thick. It is extremely friable and much of it breaks up fine enough to become suspended in the slurry water. No tests have been made to determine the exact proportion of fusain to other constituents in either the slurry or the clean coal, but some volatile analyses have been made which are significant. A hand-picked sample of fusain obtained from lumps of raw coal analyzed 14.3 pct volatile compared with 38.0 pct volatile in the lumps with the fusain removed. An analysis of a sample of the floated coal showed 23.6 pct volatile, indicating that about 61 pct of it was fusain.

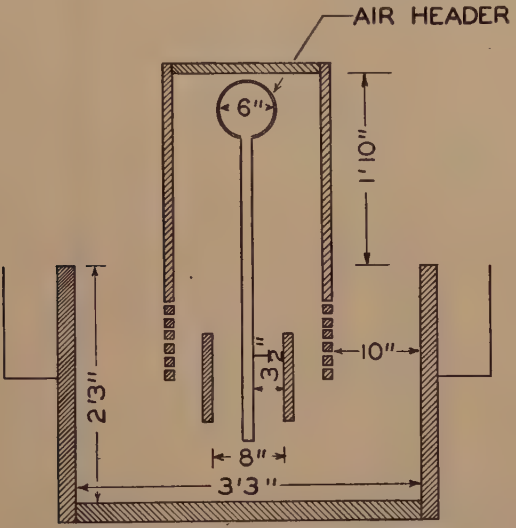

Fig 2—Cross section of 5-foot experimental pneumatic flotation machine built at the Empire Mine.

Summary

A small pneumatic flotation unit has been in operation at the Empire mine of the DeBardeleben Coal Corporation for the past 6½ yr recovering fine coal from settling-tank overflow or slurry. This water, which formerly was pumped to a settling pond, contains about 2.6 pct solids and is treated in a 12-ft Southwestern Engineering Co. pneumatic flotation cell at the rate of about 400 gpm. Ash analyses of the solids in the water average about 16.3 pct. The froth product from the pneumatic cell contains about 18.5 pct solids, which analyze on an average 4.4 pct ash. The output of the cell is about 3900 lb per hr of coal on the dry basis. This represents a recovery of about 74 pct of the total solids in the slurry feed to the flotation machine. The method of recovering and dewatering the coal in the flotation product is unusual in that the froth is flumed from the flotation machine down to the washed-coal bin, where it discharges into the bin along with the washed slack from the jigs and tables. The fine coal in the froth is recovered and dewatered by the filtering action of the bed of slack in the bin. The recovered material is high in fusain and low in sulphur and ash.

Acknowledgments

The investigation that resulted in the installation described in this paper was carried out by the DeBardeleben Coal Corporation and the U. S. Bureau of Mines in cooperation with the University of Alabama. Mr. H. T. DeBardeleben is president of the DeBardeleben Coal Corporation and Mr. H. F. DeBardeleben is executive vice president and general manager. At the time of this investigation, Dr. Milton Fies was vice president in charge of operations, and his untiring zeal and interest in the project were

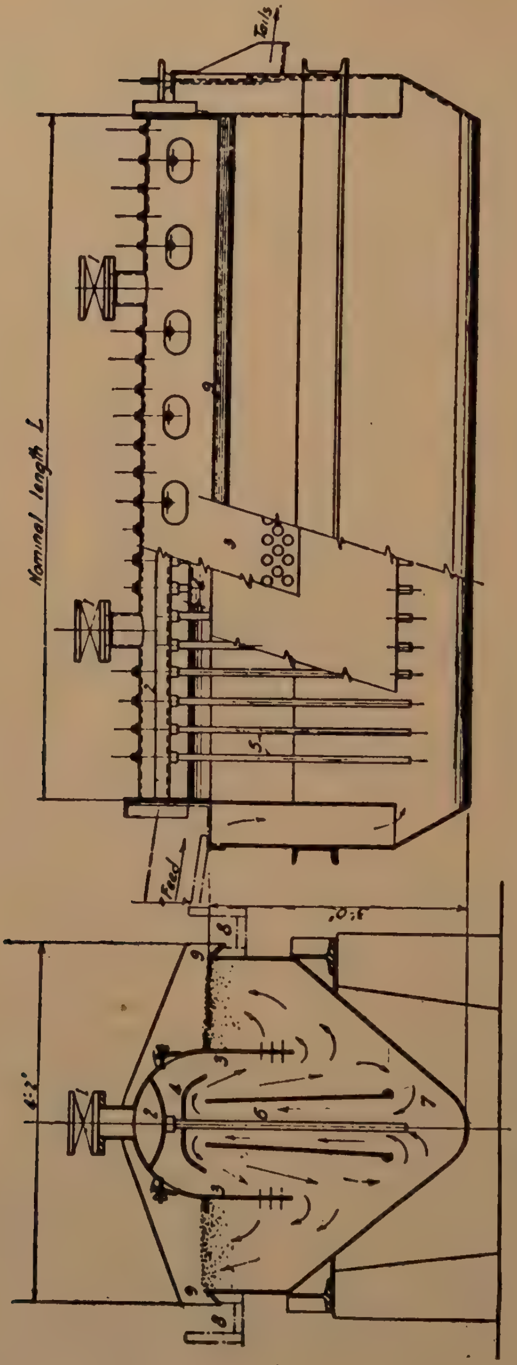

FIG 3—PNEUMATIC FLOTATION MACHINE IN USE AT EMPIRE. (*Reproduced from Richards.*[2])

largely responsible for carrying it through to the point of commercial application. The authors appreciate the valuable contributions to the investigative work made by Mr. R. G. O'Meara, formerly at the Bureau of Mines Southern Experiment Station and now metallurgist at the Bureau's Mississippi Valley Experiment Station, Rolla, Missouri.

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DISCUSSION

(D. R. Mitchell presiding)

D. R. MITCHELL*—My co-chairman arranged this symposium on Froth Flotation and, in the beginning told me that I would have to give a little preliminary discussion of the subject as well as the answers. I told him I would set up the problem, but he still insisted that I was to give some of the answers.

Froth flotation has had limited application in the United States despite a considerable amount of investigational work done during the past thirty years. At present there are four plants, two in the Pennsylvania anthracite region, one in the Pittsburgh, Pa., area, not presently operating, and one in Alabama. This condition is in sharp contrast to conditions in Europe and the British Isles where a great number of coal froth flotation plants are in operation.

The reasons for this lack of application in the United States may be summed up briefly as follows: 1. Difficulties in marketing the extreme fine sizes either alone or in admixture with coarser sizes. 2. Cost of dewatering and handling the froth flotation concentrate. 3. Indifferent test results in removing sulphur. 4. Inability of the process as presently practiced to clean successfully material finer than about 200 mesh in coal slurries. 5. Inability to clean successfully coal slurries containing a high percentage of clay.

The adoption of antistream pollution legislation in Pennsylvania and in other coal producing states, increased coal prices, and development of methods and equipment for using minus 48-mesh coal has focussed attention on the froth flotation process as a means of cleaning and recovering fine sizes formerly wasted.

As presently practiced and as proposed for new plants the general arrangement is to deslime at approximately 200-mesh, remove the approximately +35 mesh by screening, and then treat the approximately 35- × 200-mesh sizes in the flotation cells, floating the coal and depressing the impurities. Even under these conditions, particularly where clay and fine flake pyrite are present, it is often difficult to get an acceptable coal concentrate.

This is the inevitable consequence of attempting to float the major constituent (coal) with attendant mechanical entrapment of refuse particles in the usually voluminous froth. Repeated cleaning of the froth is necessary where such conditions occur resulting in a complicated and costly flow arrangement. It is believed technically possible to reverse the procedure, floating the impurities and depressing the coal. This would eliminate the difficulties attendant to handling and filtering a coal-loaded froth, and remove the necessity of passing the primary froth to cleaning cells.

There has been a slow accumulation of technical knowledge supporting these conclusions and a limited number of tests in our laboratory indicate that floating the refuse and depressing the coal is not too difficult. These facts are known: Clay and other silicate minerals are easily floated by amines and other cationic type reagents. Coal can be depressed by a number of chemicals. Stannous chloride is particularly good. Pyrite is easily floatable. Also sodium cyanide is an effective depressor of pyrite.

By the use of the contact angle method we find that oxidizing solutions such as potassium permanganate and reducing solutions such as hypophosphorus acid convert the coal surface so that a zero contact angle results with coal and an air bubble in water. Air bubbles usually give a contact angle on coal in water of 40 to 60° depending on the rank of the coal and the history of the sample under test.

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* Pennsylvania State College, State College, Pa.

What is not known is the effect of the above mentioned collectors and depressors on the other minerals present in a coal slurry and the role to be played by pH regulating reagents in such a contemplated circuit. It is believed that a detailed laboratory investigation of this problem might bear fruitful results.

It may seem a far-fetched idea to have upside-down froth flotation of coal, but considering that even though coal is very easily floatable, in most cases the major constituent is being floated, usually 60 to 80 pct coal and a small amount of impurities that are being kept down in the pulp. Technically, clay is rather easy to float and that is one constituent, particularly in bituminous coals, that causes a lot of trouble. The idea does not seem to me to be too far fetched. Probably as the session continues, we shall have some further discussion on this particular point.

J. GRIFFEN,* Co-chairman—I want to add something to what you have said. I hoped you would give some of the answers or at least explain what you have done technically in the laboratory to substantiate some of your general statements. In that way we could get some idea of the cost and economics in carrying out some of these new ideas.

My feeling about this whole thing is that we have two problems, a technical problem and an economic problem. I also feel that as coal becomes more and more valuable, something that is very fine technically today may not be economical, but next year or the year after, it may be very interesting, economically.

What I want to do is try to lay some of the technical groundwork on flotation of coal so that when the economics are ready, we shall be able to proceed.

(*J. Griffen presiding*)

J. GRIFFEN—Gentlemen, I think it has been quite interesting to find that such a simple plan can make a very satisfactory recovery, and I would now like to call for discussion.

B. M. BIRD†—Froth flotation today is assuming new importance, particularly because of the Pittsburgh area. There the steel companies are faced with a major problem in reducing sulphur, most of their low sulphur coals having been exhausted. The problem of removing pyrite did not come into Mr. Gandrud's paper but the solution is fairly straightforward, as has been demonstrated by work at Battelle Memorial Institute.

There was no difficulty in reducing the sulphur from 1.8 to about 1.25 by using cyanide and lime. Of course, these reagents increase the cost of the flotation over what it would be with Mr. Gandrud's simple process.

The logical people, of course, to take up flotation are the steel companies. I have some figures for the cost of sulphur in steel making that are based upon the work that Joseph did in 1925, but corrected for differences in labor and materials. The correction is very rough, giving a figure twice as high as Joseph's. It is 20¢ per ton of pig for each 0.05 pct sulphur.

J. GRIFFEN—Is that in coke or coal?

B. M. BIRD—That is in coke but it really comes right back to coal for practical figuring because the percentage of sulphur in the coke is about the same as in the coal. By using 1500 lb of coke per ton of pig one is back again, approximately, to the coal figure, so that in round numbers that figure applies per ton of coal.

When steel companies are spending tremendous sums of money to reduce the sulphur as little as 0.15 pct the great importance of going after the extreme fines is evident and this is a logical way to go about the problem.

I might add that one problem that looms large in the application of flotation to coal washing is dewatering. The advent of the Bird centrifugal put out by the Bird Machine Co. in Walpole, Mass., seems to offer a positive out. In other words, there is no logical reason why the froth cannot be pumped along with the balance of the ¼ × 0 right into the Bird centrifugal.

I have a question or two for Mr. Gandrud. Have you made any tests with a higher solid content in the feed to see how it would affect the cost of operation? Higher pulp densities mean smaller machines and, as a result, less power. I might say that power should not be overlooked in the operation of a flotation plant; it is one of the really big items.

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* McNally Pittsburgh Mfg. Corp., Pittsburgh, Pa.

† Jeffrey Mfg Company, Columbus, O.

B. W. Gandrud (authors' reply)—We did not make any tests to determine the possibilities of using thickeners. It is quite possible there would be a lot more capacity if the feed were thickened.

We did not make any tests because we figured the use of thickeners would involve quite a bit more expense.

H. J. Gisler*—The solids in the flotation feed is unusually low (2.6 pct) but the data presented in this paper bear out the writer's experience that flotation of fine coal requires a dilute pulp for best results.

Possibly thickening the pulp to 10 or 15 pct should be considered in this case. Greater cell capacity and lower fine oil consumption may be possible at the higher density. Has this phase of the problem been investigated at the Empire mine flotation operations?

W. J. Parton†—The development of the simple, inexpensive methods of treating washery water solids by flotation at the Empire mine of the DeBardeleben Coal Corporation is a worthy accomplishment. These developments should be thoroughly studied by any company planning the recovery of fine slurry water solids. They are of particular value to a small producer where small volumes of water and solids need be handled. Unfortunately, some of the practices developed at this particular plant appear to be impractical for larger installations.

Several significant developments were discussed in this paper:

1. The dilute waste water carrying approximately 2.5 pct by weight solids was treated by feeding it directly to the flotation cells. This practice is possible where small volumes of water are discharged from the cleaning plant and in cases where preliminary desliming is not required to remove colloidal clay particles. Our experience with the flotation of fine coal of the size treated at the Empire mine has been that efficient separation in relatively short flotation contact time is possible in the flotation of dilute pulps. However, when dilute pulps are handled, larger cell capacity with consequent increased power consumption is necessary to obtain the contact time required to give satisfactory recoveries. For large installations, thickening of the slurry solids before flotation to reduce the volume would probably be more economical in the long run because of a reduced number of cells and power.

2. The use of air cells for flotation of coal is of particular interest since most coal plants employing flotation have used the mechanical cell. However, the air cell has several advantages, such as: simplicity, low initial cost and low operating and maintenance cost. A new flotation plant recently placed in operation in the anthracite region is the first large plant on coal in this country using the air cell of the Steffensen design. Metallurgical results for the two types of cells should be compared to serve as a guide in the design of future fine-coal recovery plants.

3. Dewatering of the coal froth by discharging it directly to the slack bin (where the excess water drains through the bed of coal serving as a filter medium) is a simple way of solving the most difficult problem in recovering fine coal. This method requires no mechanical dewatering equipment and entails no extra operating cost. Unfortunately, this practice probably will be limited for use where only small volumes of froth are handled. In large flotation plants with 30 to 40 tons per hr of coal carried in 250 to 400 gpm of froth, the use of a slack bin for dewatering appears impractical. Some preliminary mechanical dewatering equipment would be necessary before blending the coal sludge or cake with the slack.

4. No mention is made of the use of conditioning the raw feed pulp with the reagent before it enters the cells. Some premixing of the reagent with the feed pulp before flotation would most likely result in higher coal recoveries with a reduced consumption of pine oil. The use of a small proportion of fuel oil or kerosene with the pine oil would undoubtedly further reduce pine oil requirements. However, it is likely that the use of oil was found to be objectionable at this operation due to an increase in the ash content of the product.

B. W. Gandrud—We do not have any

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* Denver Equipment Co., Denver, Colo.

† Nesquehoning Colliery, Lehigh Navigation Coal Co., Inc.

conditioning except that the raw slurry discharges into a big barrel before it goes into a machine and the pine oil also discharges into this barrel. There is much agitation in the barrel and this is equivalent to some conditioning.

I also want to say that I thoroughly agree with you that this method of recovering the fine sizes by flotation has very definite limitations. It is not an unmixed blessing at all and, as you say, if too much of the floated material was run into the washed coal bin, there would be trouble immediately. No doubt it adds a little more moisture to the slack coal so that it will have a little higher moisture content after it has been loaded into the railroad cars. It might not work out well in the northern plants, at least in the wintertime, because of the possibility of freezing. It has very definite limitations. Further south, there can be an amount of water within 30 or 40 gpm if a fairly good bed is kept in the slack coal bin.

B. M. Bird—Mr. Gandrud will you discuss clay? Knowing the Black Creek coal region, I imagine you had something bordering on white paint there.

B. W. Gandrud—No, the clay is not particularly noticeable. If we tried to deslime, I do not think it would work. We should probably lose most of the 200-mesh and consequently most of the coal. But the clay is not at all conspicuous in that plant now. It may have been, some years ago when you were there, but it is scarcely noticeable now.

B. M. Bird—This is not an open and shut answer to the problem of floating in the presence of clay.

B. W. Gandrud—I doubt that the clay would interfere. It might but I do not know about that.

D. H. Davis*—This paper is particularly interesting to me because it presents results with a pneumatic flotation machine whereas in the other two papers of this session, the results of mechanically agitated machines were the subject of the papers.

The feed pulp is extremely dilute, 2.6 pct solids, and no doubt this is the reason for such a clean concentrate. The ash reduction is very good and likewise the tailings are reasonably high in ash content. More data on various mesh sizes would be of interest as well as some data on a balance of solids and water in respect to gallonage or tonnage.

It appears to the writer that the most significant point presented is the increase in solids content from 2.6 pct in the feed pulp to 18.5 pct in the froth concentrate. This would indicate an exceptional separation of the froth from the pulp. At 18.5 pct solids a filter may be operated but not at a very high capacity. The writer was privileged to observe the operation of this unit several years ago; the action and performance of the pneumatic Forrester type cell appeared to be excellent, particularly under the conditions of such a low per cent solids in the feed.

The authors are to be commended for presenting this interesting paper on flotation in Alabama.

W. L. Remick*—I would like to add something to that clay discussion. Just twenty years ago I ran a flotation test at the Bradford mine on the Black Creek coal. That was the entire range of sludge from 10 in. to 0 in size. There was so much clay that the water drained down through 16 ft of coal in the bin and came out like milk. As the coal was floated, the clay was entirely rejected. It made a froth under 3 pct ash and apparently rejected the clay. Had it been any other kind of clay, it might be a different story.

S. C. Sun†—I think it is interesting that in your concentrate you have about 60 pct of fusain because fusain has been considered as one of the undesirable petrographic constituents in the process of coal hydrogenation. The finding of a simple process for the removal of fusain from coal may contribute something to the field of coal hydrogenation.

At the Pennsylvania State College, I am carrying on the research work of coal flotation and particularly on the removal of fusain from coal. However, the final conclusion has not been formulated as yet.

I wish to ask you one question. Do you think that fusain has a better floatability than the other petrographic constituents such as clarain, vitrain, and durain? Furthermore, do you have any estimation of the content of fusain in your feed?

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* Pittsburgh Coal Co., Library, Pa.

* Roberts and Schaeffer Co., Chicago, Ill.

† Pennsylvania State College, State College, Pa.

B. W. Gandrud—I am not positive about this, but I think the fusain is a whole lot finer in size and because of this it tends to float better in a pneumatic machine than the other sizes. That is probably one reason that the fusain is concentrated in the floated product. Just how much fusain there would be in the feed, I do not know. I do not recall what is the analysis, but there is a lot of fusain in the feed—probably 40 to 50 pct.

S. C. Sun—From the data of my research work, it would appear that the floatability of fusain in distilled water without the addition of activators and chemical collectors, is somewhat lower than that of the other petrographic constituents of coal. Although my research work is still in progress and my data may not be sufficient to draw a conclusion I think the conception that more fusain in the concentrate may be caused by some other factor than floatability is probably right. As to the other point concerning the amount of coal in the flotation pulp, my laboratory data indicated that the pulp consists of 10 up to 15 pct of coal by weight.

Mr. Bird mentioned something about the use of lime and cyanide for the depressing of pyrite. I should like to know what pH value was used.

B. M. Bird—I cannot answer that definitely. It was a little above 7, I think.

S. C. Sun—I think it should be, because pyrite floats better on the acid circuit. Therefore, this brings us back to the point mentioned by Professor Mitchell. Since the amount of pyrite presented in most coal is rather small, we should try to float the pyrite instead of coal.

Froth Flotation of Minus 48-mesh Bituminous Coal Slurries

By D. H. Davis,* Member AIME

(New York Meeting, March 1947)

There are two general types of oil flotation processes, froth and bulk-oil. In froth flotation, the coal concentrates are removed in the form of a froth or foam composed of air, liquid, and solids. Mechanical or pneumatic agitation of a coal slurry with suitable frothing agent causes a selective action of the bubbles to the coal particles which are carried to the upper portion of the pulp in the flotation cell; the refuse is wetted by the water and does not rise with the froth, but goes out as tailings. The other type of flotation process, bulk-oil, consists of thoroughly mixing the slurry or pulp with 30 to 50 pct of its weight of oil. The coal agglomerates with the oil and drops to the bottom of the cell while the refuse is wetted by the water and is carried away in suspension. The concentrate granules may be screened and drained to 12 to 15 pct moisture. The minus 200 or 300-mesh is very effectively treated by this process. The Trent process is a bulk-oil type which has been used for coal cleaning on a commercial basis, but its use has been discontinued in this country.

Froth flotation of bituminous coal should be subdivided into two distinct fields: (1) separation of a dry coal product such as recovered from an aspiration system; (2) separation of a minus 48-mesh slurry such as produced from a thickener in a wet cleaning plant. The capacities, performance, and general operating characteristics are very different in these two fields. The recirculated slurry in a washery becomes oxidized and is much harder to treat than a dry dust. It is difficult to clean a slurry containing an appreciable percentage of plus 48-mesh without also floating some high-ash clays minus 200-mesh in size. On the other hand, while working with an aspirated dust, good results have been obtained in floating coal as coarse as 3/16-in. Because of the relatively low capacity of flotation machines as compared with conventional fine-coal cleaners and because of the greater quantity of reagents required to float coarse coal, there is usually no point in considering flotation on bituminous coal for anything coarser than approximately 48-mesh. In some coal-cleaning plants, good separations are being made down to 80 or even 100-mesh by a less expensive system of coal cleaning than flotation.

The Pittsburgh Coal Co. carried out extensive tests on laboratory batch machines and on small units of both pneumatic and mechanical agitation types from 1930 to 1935 and have operated six, eight, and then twelve mill-size mechanical subaeration cells from 1932 to 1944. This paper will deal with some of the operating phases and results of coal-flotation practices as derived from this experience, and more specifically with the flotation practice at the Champion No. 1 preparation plant. The experience gained from the research and initial experimental work contributed greatly to the successful operation of the mill-sized flotation units at Champion.

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* Product Control Manager, Pittsburgh Coal Co., Library, Pennsylvania.

Water-settling System of Washery

In the Rheolaveur system used at Champion, all the minus ⅜-in. coal with the water used in the coarse and fine coal cleaning circuits enters a series of elevators boots, which make a size classification of the solids at approximately 48-mesh. Part of the overflow from the boots is returned to the Rheolaveur circuit for coal cleaning, but approximately 30 pct is continually diverted to a system of thickeners for classification of the solids and for recovery of clarified water.

Fig 1 shows a flowsheet of the thickeners and flotation plant. Feed to the thickeners enters No. 1 thickener (60-ft diam); a portion of the overflow is returned to the clarified-water sump for reuse in the Rheolaveur plant, while the remainder of the overflow is distributed between No. 2 thickener (35-ft diam), and No. 3 thickener (85-ft diam). The solids that settle in these thickeners are removed from the underflow pipes as a slurry containing from 35 to 45 pct solids.

Table 1 shows typical data on the operation and products of these three thickeners. No. 1 thickener is being operated as a classifier with a feed of 4000 gpm from the ⅜-in. to 0 coal boots' overflow. Overflow from the No. 1 thickener is divided as follows: 250 gpm is fed into the No. 2 thickener, 600 gpm goes to the No. 3 thickener and the remainder is returned to the plant clarified-water sump. In addition to the 600 gpm from the No. 1 thickener overflow, No. 3 thickener receives 150 gpm from a ⅜-in. to 0 refuse boot overflow and 350 gpm of flotation tailings. The overflow water of the No. 2 and 3 thickeners is clear containing practically no solids. The operation of No. 1 thickener as a classifier with a high gallonage of feed results in the No. 1 thickener underflow being the coarsest product, that is, it contains the least percentage of minus 200-mesh, and is the lowest in ash content (14.1 pct ash). No. 2 thickener underflow contains 25 pct ash and is the lowest in tonnage of solids. The underflow of No. 3 thickener contains the highest percentage of minus 200-mesh in which colloidal clay containing over 30 pct ash is concentrated.

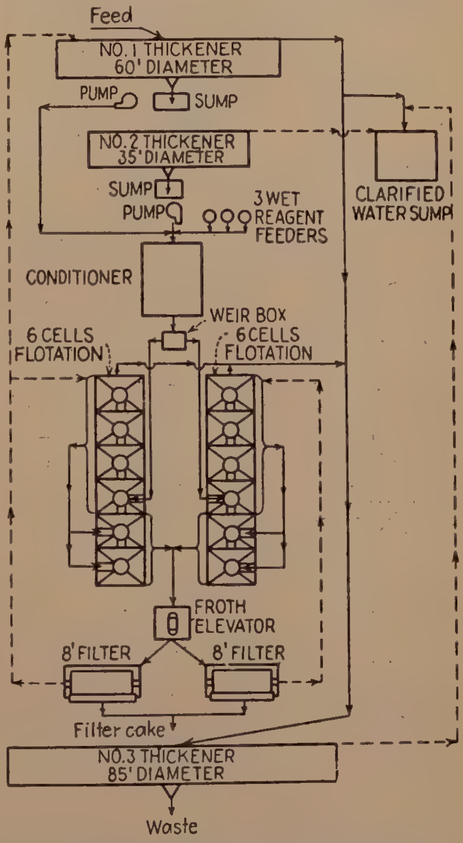

Fig 1 —Flowsheet of Froth-flotation Units and Water System at Champion No. 1 Preparation Plant.
Feed-portion of Overflow from Fine Coal (⅜ × 0 in.) Boots.

Flotation Feed

Usually the feed to the flotation cells consists of either all or part of the combined underflows from No. 1 and No. 2 thickeners. Normal practice is to send the tailings from the flotation plant to No. 3 thickener, but at various intervals the tailings have been sent to No. 2 thickener and the underflow from No. 2 sent to waste. By such an arrangement, it was possible to feed the underflow of No. 3 thickener to flotation, thereby cleaning in flotation a product of high ash content and consisting of a high percentage of minus 200-mesh. By such variations of the water circuit, it has been possible to operate flotation on feeds of various characteristics.

TABLE 1—*Typical Data on Thickener Operation, Sieve Analyses and Ash of Thickener Underflows*

Description or Tyler Mesh	Feed		Overflow		Underflow	
	Wt Pct	Ash	Wt Pct	Ash	Wt Pct	Ash
No. 1 Thickener (60 Ft Diam) Feed—4000 gpm From ⅜ in. to 0 Coal Elevator Boots' Overflow						
					(To Flotation)	
Head	100.0	21.6	100.0	22.3		14.1
On 48 mesh	Trace				8.0	5.1
On 100 mesh	8.0	5.9	2.0	7.1	38.0	6.2
On 200 mesh	12.0	10.1	12.0	7.6	27.0	13.1
Thru 200 mesh	80.0	24.9	86.0	24.7	27.0	30.6
Pct solids	8.2		6.7		37.8	
gpm	4000		3792		208	
tons per hr					21.8	
No. 2 Thickener (35 Ft Diam) Feed—250 gpm of No. 1 Thickener Overflow						
					(To Flotation)	
Head	100.0	22.7			100.0	23.9
On 48 mesh			Clear Water			
On 100 mesh						
On 200 mesh	8.0	7.6			8.0	8.6
Thru 200 mesh	92.0	24.7			92.0	25.0
Pct solids	6.7		0.0		29.9	
gpm	250		218		32	
tons per hr					2.6	
No. 3 Thickener (85 Ft Diam) Feed—600 gpm 60 Ft Thickener Overflow, 150 gpm ⅜ in. to 0 Refuse Boot Overflow, 350 gpm Flotation Tailings						
					(To Waste)	
Head					100.	32.7
On 48 mesh	Not Sampled					
On 100 mesh			Clear Water		4.	15.6
On 200 mesh					10.	24.8
Thru 200 mesh					86.	34.3
Pct solids	6.7		0.0		39.4	
gpm	1100		1016		84.	
tons per hr					9.2	

VARIATION OF FLOTATION FEED

Coal washers are not operated continuously over a 24-hr period and because of this discontinuous process there is a wide variation in the density of thickener underflow pulps and the size consist of the solids throughout the operating hours. This variation in the pulp density and size consist of the solids takes place from hour to hour during each day and from day to day during the week and is caused mostly by the intermittent rather than continuous operation of the thickeners. It is also affected by the operation of the coal-cleaning and water-settling equipment such as the amount and variation of drawoff from the thickeners, build-up of solids in the washery water that results in a coarser feed to the thickeners, and the method of slurry withdrawal and disposal. All these factors influence the type of product produced in and removed from the thickeners. The conditioner ahead of the flotation cells serves as a surge tank for evening out some of the variations in the flow of slurry from the thickeners to the flotation units in addition to its prime purpose of providing a means for intimate mixing of the flotation feed with flotation reagents.

FUNCTION OF REAGENTS

Reagents are added to the feed pulp and dilution water ahead of the conditioning tank. Careful regulation and control of reagents are most important and this is readily accomplished by the use of bucket-type reagent feeders.

A brief summary of the various classes of flotation reagents may be of interest. Frothing agents are used for the production of foam without which no flotation is possible. Pine oil, cresylic acid, anilin, toluidine, and synthetic alcohols are examples of frothers.

Collecting agents are used to gather certain minerals in the foam and sometimes react to form an unwettable surface causing adherence to air bubbles. Fatty acids and soaps such as xanthates are examples in mineral flotation. In coal flotation, water gas tar, spray oil, and fuel oil are the most common collectors used.

Depressing agents are used to retard or prevent the collection of certain minerals; sulphites, cyanides, alkali silicates, sulphates, and lime are examples. In coal flotation pyrite depression is the most important use for depressants.

Glues and starches known as protective colloids have a strongly depressing action, being absorbed by the mineral particles which show no tendency to float. In some cases, dispersing reagents are used to release oils and disperse the pulp. Agents such as sodium carbonate, sodium silicate, sodium cyanide are used for dispersion.

CONDITIONING THE FEED

At Champion the underflow from the thickener or thickeners that is to be cleaned by flotation is pumped at a pulp density from 35 to 45 pct solids to the 2000-gal cylindrical conditioning tank operating on the "hydrotator" principle. The slurry, dilution water, and flotation reagents are fed together into a small tank and then into the conditioning tank. Intimate contact and mixing is accomplished by the circulation of the pulp through the pump, pipes, and jets of the conditioning tank. The conditioner is of sufficient capacity to permit about 5 or 6 min retention time for the physical and chemical reactions to take place between the reagents and the pulp. The conditioner also acts as an equalizer between the thickener and the flotation units.

FLOTATION CELLS AND CIRCUIT

The laboratory investigations of flotation which preceded the operating phase established the basic flowsheet for the mill flotation circuit and indicated that retreatment of a primary concentrate is necessary in order to make a reasonably good separation between coal and refuse. In the experience of the Pittsburgh Coal Co., it has been practically impossible to effect a good separation with respect to both coal and refuse on one pass through the machine, just the same as in many other coal cleaning processes for fine coal. Also, it has been found advisable to reclean the middlings from the cleaner cells as these middlings are too low in ash to discard as refuse.

The flotation plant at Champion consists of two banks of six cells operating in parallel, each bank being operated as one unit independent of the other bank. The cells used are of the mechanically agitated subaeration type, 43 by 43 in. in cross section at the bottom of the cell with concentrate discharge on two sides of the cell. A multicell arrangement is possible with this type of flotation machine and concentrates from rougher cells may be flumed so as to enter cleaner cells through middling return openings. The multicell arrangement of the cells is shown by the flowsheet (Fig 1). Feed enters the third cell and cells No. 3, 4, 5, and 6 are operated as "rougher" cells producing a primary concentrate. The primary concentrate is diluted with filtrate water and flows by gravity to cells No. 1 and 2 where the primary concentrate is recleaned to produce a final concentrate. Cells No. 1 and 2 are designated as "cleaner" cells. The middlings produced from No. 2 cleaner cell joins the primary flotation feed to No. 3 cell. The refuse passes progressively through cells No. 3, 4, 5, and 6, where it is discharged as a final tailings. Several combinations of "rougher" and "cleaner" cells were tried in the experimental work with small cells, but the flowsheet described was found to give the best results at the greatest capacity. One bank of cells is numbered 1 to 6 inclusive and the other bank 7 to 12 inclusive; in the latter bank, the feed enters No. 9 cell, primary concentrate is produced from cell No. 9, 10, 11, and 12, and recleaned in cells No. 7 and 8. Cells No. 7 and 8 produce a final concentrate and a final tailing is produced from No. 12 cell.

TABLE 2—*Summary Data on A Series of Tests with Various Thickener Underflows as Flotation Feed*

Test No.	Feed		Final Concentrates		Final Tails		Recovery, Pct	Solids, Pct			Gallons per Minute				Dry Tons per Hr		
	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	(Ash Basis)	Feed	Final Conc.	Final Tails	Feed	Final Conc.	Final Tails	Dilution H_2O	Feed	Final Conc.	Final Tails
No. 1 Thickener Underflow—Flotation Feed																	
1	12.5	2.7	7.4	2.2	48.1	4.8	87.5	24.1	27.7	6.0							
2[1]	12.1		6.4		51.7		87.4	22.0	39.6	2.4							
3	12.6	2.7	7.6	2.4	72.0	5.6	92.2	31.5	36.5	3.1	204	162	180	138	17.5	16.1	1.4
4	13.1		7.6		52.1		87.6	36.2	36.9	6.3	187	161	152	126	18.8	16.4	2.4
Nos. 1 and 2 Thickener Underflows—Flotation Feed																	
5	14.7	2.7	7.5	2.2	78.8	5.9	89.6	18.3	33.7	1.8	446	208	486	248	21.4	19.2	2.2
6	13.6	3.0	7.8	2.4	67.0	6.3	90.1	25.2	38.1	3.0	296	202	292	198	22.3	20.1	2.2
7	14.6	2.6	7.1	2.2	48.1	4.6	81.7	21.5	34.7	4.4	319	164	305	150	18.6	15.2	3.4
8	15.7	2.6	7.8	2.3	48.2	4.4	80.5	20.1	31.8	5.1	412	203	324	115	21.8	17.6	4.2
9	14.7	2.6	7.6	2.3	52.8	4.7	84.3	20.6	32.1	4.0	402	212	335	145	22.0	18.6	3.4
10	15.2	2.7	8.4	2.4	72.4	5.4	89.4	22.4	31.9	3.6	420	266	287	133	24.8	22.2	2.6
11	14.7	2.7	8.0	2.3	66.2	5.1	88.5	22.9	32.5	3.9	388	244	274	130	23.8	21.1	2.7
12	14.5	2.6	7.6	2.2	71.3	5.0	89.1	20.6	29.3	2.6	384	241	321	178	20.8	18.6	2.2
13	14.6	2.6	7.2	2.3	54.0	4.6	84.2	22.2	32.0	4.2	354	202	314	162	20.9	17.6	3.3
14	15.0	2.7	8.8	2.3	64.8	5.7	86.4	22.0	32.9	4.3	352	197	257	102	20.6	17.8	2.8
No. 3 Thickener Underflow—Flotation Feed																	
15	25.0		8.7		50.1		60.5	18.3	20.7	6.0	242	127	298	183	11.6	7.0	4.6
16[1]	23.2		11.1		72.2		80.2	19.3	17.1	4.8							

[1] Six cells only being operated.

OPERATION AND RESULTS OBTAINED BY FLOTATION

J. T. Crawford[1] stressed the importance of maintaining a dilute pulp in the cell in order to obtain the best separation and found that "where the actual condition in the cells is based on from 10 to 12 pct solids (by weight), the best results are achieved in the flotation cell." Operating experience has indicated that 18 pct solids is about the maximum desirable pulp density when operating on a feed with 25 to 30 pct minus 200-mesh and most of the remaining solids between 48 and 200-mesh. As a result of the addition of dilution water to the primary concentrates and to the recirculation of middlings from the secondary cells back to the primary cells, the actual feed to the primary cells is of lower density than indicated by the percentage of solids of the primary flotation feed. Also due to the concentrates containing about 30 pct solids, the pulp density in the cell is considerably lower than in the primary feed.

In operating the flotation plant at

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[1] J.T. Crawford: Importance of Pulp Density, Particle Size and Feed Regulation in Flotation of Coal. *Trans.* AIME (1936) **119**.

TABLE 3—*Reagents Used in Tests of Table 2*

Test No.	Reagents—Lb per Ton of Feed									
	Frothers				Collectors			Depressants		
	Cresylic Acid	Tar Acid Oil	Synthetic Alcohol	Pine Oil	Spray Oil	Fuel Oil	Kerosene	Dry Lime	Ferrous Sulphate	Ferric Sulphate
	No. 1 Thickener Underflow—Flotation Feed									
1	0.38					1.0				
2	0.40					1.3				
3	0.45				0.90		0.30			
4	0.36				0.78		0.27			
	Nos. 1 and 2 Thickener Underflows—Flotation Feed									
5	0.42				0.52		0.18			
6	0.51				0.68		0.23			
7	0.40				0.60		0.20	2.0		
8	0.40				0.75		0.25	2.0		
9			0.24		0.60		0.20		0.50	
10	0.37				0.60		0.20			
11	0.45				0.60		0.20			0.48
12	0.43				1.03		0.34			
13		1.48			1.20		0.40			
14				0.45		1.20				
	No. 3 Thickener Underflow—Flotation Feed									
15	0.40					1.60				
16	0.60					1.80				

TABLE 4—*Analyses of Products, Flotation Test No.* 5
Feed: Nos. 1 and 2 Thickener Underflows

Products	Head			Sieve Analyses—Tyler Mesh								
				On 48 M		48 to 100M		100 to 200 M		Minus 200 M		
	Solids Pct	Ash	Sulphur	Wt Pct	Ash	Wt Pct	Ash	Wt Pct	Ash	Wt Pct	Ash	Sulphur
Cell Nos. 1, 2, 3, 4, 5, 6												
Primary feed (to No. 3 cell)	18.3	14.7	2.7	16.5	8.6	34.5	7.7	21.0	15.2	28.0	29.8	4.9
Middlings (No. 2 to No. 3 cell)	2.0	44.8	4.3	9.5	34.0	31.5	36.7	15.5	35.4	43.5	59.1	6.5
Primary concentrates (No. 3–4 cells)	34.6	8.9	2.4	18.0	4.1	35.0	6.0	21.0	10.1	26.0	16.0	4.1
Primary concentrates reclean No. 5–6 cells)	32.7	15.3	3.8	16.0	5.6	26.5	7.6	18.0	17.2	39.5	25.7	6.3
Secondary concentrates (No. 1–2 cells)	33.5	7.6	2.3	17.0	4.5	34.5	5.6	22.0	8.5	26.5	12.2	3.5
Tailings (No. 6 cell)	1.7	79.8	5.6	6.5	65.1	34.0	78.8	15.5	79.8	44.0	72.8	6.7
Cell Nos. 7, 8, 9, 10, 11, 12												
Primary feed (to No. 9 cell)	18.3	14.7	2.7	16.5	8.6	34.5	7.7	21.0	15.2	28.0	29.8	4.9
Middlings (No. 8 to No. 9 cell)	1.8	34.3	4.2	10.0	7.4	35.0	23.2	20.0	31.2	35.0	53.5	6.7
Primary concentrates (No. 9–10 cells)	37.2	8.6	2.3	20.0	4.5	35.5	6.0	20.5	10.1	24.0	14.8	3.8
Primary concentrates reclean (No. 11–12 cells)	32.2	14.7	3.6	16.5	6.0	25.0	8.1	18.0	18.7	40.5	21.2	5.6
Secondary concentrates (No. 7–8 cells)	34.0	7.4	2.1	19.0	4.5	35.0	5.7	20.0	8.8	26.0	12.0	3.5
Tailings No. 12 cell)	1.9	77.9	6.2	2.0	60.2	29.0	75.8	17.0	77.9	52.0	80.4	7.3

Circuit:
Two six-cell units operating in parallel. Feed to No. 3 and No. 9 cells. Froth from No. 3, No. 4, No. 5, and No. 6 recleaned in No. 1 and No. 2. Froth from No. 9, No. 10, No. 11, and No. 12 recleaned in No. 7 and No. 8. Secondary concentrates (final cleaned coal) from No. 1, No. 2, No. 7, No. 8. Tailings (final refuse) from No. 6 and No. 12.

Champion, it was of great importance to maintain a final concentrate of sufficiently high percentage of solids that it could be sent directly to filters without thickening as there was no thickener available or a suitable location for one between the concentrate launders and the filters. Consequently, it was often impossible to operate the flotation units with a feed at the optimum density for best technical separation results. It was necessary rather to operate at such a density that the flotation concentrates could be filtered without too many difficulties and under this condition obtain the best possible cleaning. Thus a compromise in respect to density of the feed and filtering capacity had to be made to accomplish best overall results. It is very probable that better cleaning results could have been obtained if it were possible to operate at a lower percentage of solids in the feed in order to maintain the best density conditions for separation in the cells.

Table 5—*Balance of Solids and Gallonage, Flotation Test No. 5*

Products	Gpm	Solids, Pct	Solids		
			Tons per Hr	Feed, Pct	Ash, Pct
Rougher Circuit (Both Units)					
Primary feed (to No. 3 and No. 9 cells)	446	18.3	21.4	100.0	14.7
Middlings (No. 2 to No. 3, No. 8 to No. 9 cells)	263	1.9	1.3	6.1	44.8
Actual feed to No. 3 and No. 9 cells)	709	12.3	22.7	106.1	16.7
Primary concentrates (No. 3, 4, 5, 6 cells)	223	33.5	20.5	95.8	10.3
Tailings (No. 6, No. 12 cells)	486	1.8	2.2	10.3	78.8
Cleaner Circuit (Both Units)					
Primary concentrates (No. 3, 4, 5, 6 cells)	223	33.5	20.5	95.8	10.3
Dilution water to (No. 1, 2, 7, 8 cells)	248				
Actual feed to (No. 1 and No. 7 cells)	471	16.6	20.5	95.8	10.3
Secondary concentrates (No. 1, 2, 7, 8 cells)	208	33.7	19.2	89.7	7.5
Middlings (No. 2, 8 cells)	263	1.9	1.3	6.1	44.8

Reagents:
0.42 lb per ton cresylic acid.
0.18 lb per ton kerosene.
0.52 lb per ton coal spray oil.
Circuit:
See Table 4.

Results for three general types of feeds are available: (1) underflow from No. 1 thickener; (2) underflow from No. 1 and No. 2 thickeners and; (3) underflow from No. 3 thickener. In the latter case, the flotation tailings were sent to No. 2 thickener, which received no feed except the flotation tailings. Table 2 gives a summary of data on a series of tests covering each of the three types of feeds and is presented to show the cleaning performance that has been obtained in day to day performance of regular mill operation. Table 3 gives the reagent data for the corresponding tests of Table 2.

In order to examine in detail the results that have been obtained, complete test information on test No. 5 is presented in Tables 4 and 5. The results of this test are considered quite satisfactory; with a feed of 21.4 tons per hr at 18.3 pct solids, a secondary concentrate of 7.5 pct ash and 2.2 pct sulphur and a tailings of 78.8 pct ash and 5.9 pct sulphur were produced at a recovery of 89.6 pct. These results are attributed to the low pulp density in the cells where the separation between the various types of particles actually takes place. The dilution water added to the primary concentrates ahead of the cleaner cells was 248 gpm, the highest dilution of any of the tests. The secondary concentrates contained 33.7 pct solids and handled satisfactorily in the vacuum filters; the final tailings were very dilute at 1.8 pct solids. A sulphur reduction from 2.7 pct in the primary feed to 2.2 pct in the secondary concentrates and the high-ash tailings (78.8 pct ash) in conjunction with the 7.5 pct ash in the concentrate is to be noted. The actual feed to the rougher cells was 12.3 pct solids and to the cleaner cells 16.6 pct solids.

The results of test No. 15 on the No. 3 thickener underflow are shown in Table 6.

TABLE 6—*Analyses of Products, Flotation Test No. 15*
Feed: No. 3 Thickener Underflow

Products	Solids, Pct	Head Ash	Sieve Analyses—Tyler Mesh							
			On 48 M		48 to 100 M		100 to 200 M		Minus 200 M	
			Wt Pct	Ash	Wt Pct	Ash	Wt Pct	Ash	Wt Pct	Ash
Primary feed (to No. 3 and 9 cells)	18.3	25.0	7.5	22.8	5.5	22.3	14.5	16.8	72.5	27.8
Middlings (No. 2 to 3 and No. 8 to 9 cells)	6.3	18.8	3.5	16.5	8.5	17.3	13.5	12.1	74.5	20.6
Primary concentrates (No. 3, 4, 9, 10 cells)	17.3	12.7	1.0	8.6	4.0	11.4	43.5	11.6	51.5	14.4
Primary concentrates reclean (No. 5, 6, 11, 12 cells)	21.3	12.6	3.0	6.1	5.5	10.0	20.5	8.6	71.0	14.6
Secondary concentrates (No. 1, 2, 7, 8 cells)	20.7	8.7	1.0	7.8	2.5	8.0	52.0	7.9	44.5	9.1
Tailings (No. 6 and 12 cells)	6.0	50.1	38.5	50.2	9.0	48.1	9.0	37.9	43.5	50.3

Balance of Solids and Gallonage
Rougher Circuit

	Gpm	Solids, Pct	Solids			Reagents
			Tons per Hr	Feed, Pct	Ash, Pct	
Primary feed (to No. 3 and 9 cells)	242	18.3	11.6	100.0	25.0	0.40 lb per ton cresylic acid
Middlings (No. 2 to 3, No. 8 to 9 cells)	278	6.3	4.5	38.8	18.8	1.60 lb per ton No. 2 Fue
Actual feed to (No. 3 and 9 cells)	520	12.0	16.1	138.8	23.2	
Primary concentrates (No. 3, 4, 5, 6, 9, 10, 11, 12)	220	19.3	11.5	99.1	12.7	
Tailings (No. 6 and 12 cells)	300	6.0	4.6	39.7	50.1	
Cleaner Circuit						
Primary concentrates (No. 3, 4, 5, 6, 9, 10, 11, 12)	220	19.3	11.5	99.1	12.7	
Dilution water (to No. 1, 2, 7, 8 cells)	184					
Actual feed to (No. 1 and 7 cells)	404	11.0	11.5	99.1	12.7	
Secondary concentrates (No. 1, 2, 7, 8 cells)	126	20.7	7.0	60.3	8.7	
Middlings (No. 2 and 8 cells)	278	6.3	4.5	38.8	18.8	

Circuit: Two six-cell units operating in parallel. Feed to No. 3 and No. 9 cells. Froth from No. 3, No. 4, No. 5, and No. 6 recleaned in No. 1 and No. 2. Froth from No. 9, No. 10, No. 11, and No. 12 recleaned in No. 7 and No. 8. Secondary concentrates (final cleaned coal) from No. 1, No. 2, No. 7, and No. 8. Tailings (final refuse) from No. 6 and No. 12.

The primary flotation feed had a head ash of 25.0 pct and contained 72.5 pct minus 200-mesh at 27.8 pct ash, indicating that this product was high in clay. Because of the high percentage of minus 200-mesh in the No. 3 thickener underflow, the feed was regulated so as not to exceed one ton per hour per cell and the feed during this test was 11.6 tons per hr at 18.3 solids. An exceptionally good separation was made on the minus 200-mesh, the reduction in ash being from 27.8 to 9.1 pct with a tailing of 50.3 pct ash. The actual feed to Nos. 3 and 9 cells (rougher) was 12 pct solids and to No. 1 and 7 cells (cleaner) 11 pct. The low pulp density and proper feed tonnage are accountable for the good results on the minus 200-mesh. The source of the plus 48-mesh in the feed is the refuse boot overflow, which is sent to the No. 3 thickener because of its high ash content. It is to be noted that the separation on the 100 to 200-mesh size was the most incomplete, the ash of the tails on this size being but 37.9 pct as compared with about 50 pct ash on the other sieve sizes. Because of the marketing and drying problem of such an extremely fine product, it was usually necessary to send the underflow of No. 3 thickener to waste, and little opportunity was afforded to operate with it as feed to flotation.

Factors Affecting Operation of Flotation Units

The effect of variation in the characteristics of feed has been previously mentioned but it is of sufficient importance to be emphasized again. Variation in the percentage of solids and in the size consist of the solids requires varying amounts of reagents. For instance, as the amount of plus 48-mesh increases, an increased quantity of reagent is needed to float the larger size of material and this causes some high-ash fines to be floated that are not desired in the concentrate. The more minus 200-mesh clay there is in the feed, the more difficult it becomes to make a sharp separation, and also the less is the capacity of the cells.

With a high percentage of plus 48-mesh coal in the feed, there is created an extremely unequal distribution of sizes on which flotation must operate, resulting in a strong tendency for plus 48-mesh to go out with the tailings; for this reason plus 48-mesh should not be considered as belonging to a normal flotation feed.

Because of the high percentage of ash in the minus 200-mesh and the low percentage of ash in the plus 48-mesh, the problem becomes one of either cleaning the fines with a loss of some coarse material or floating the coarse coal accompanied by some high-ash fines. Minus 200-mesh requires smaller quantities of reagents than coarse such as, plus 48-mesh with the result that an over oiling of the fines takes place. Dilute pulps require more reagents than pulps of higher percentages of solids. To obtain as good cleaning on an unsized feed as on a closely sized feed requires a more dilute pulp density, more recleaning of the concentrates, and less tonnage per cell.

The optimum pulp density for flotation feed is dependent more on the percentage of minus 200-mesh than on the ash content of the product. For example, with 30 pct minus 200-mesh in the feed, a pulp density of 18 to 20 pct may be satisfactory, but with 80 pct minus 200-mesh 12 to 15 pct solids will probably be necessary to give an equivalent cleaning performance. With a high percentage of solids in the feed such as 30 pct or higher, a considerable amount of plus 48-mesh coal usually goes out with tailings.

Float and sink tests frequently do not give accurate results on the washability characteristics of fines under 100-mesh and flotation is the best means of determining ash and sulphur elimination. In laboratory tests, the best reduction of impurities was made in the minus 200-mesh; in one instance, 84.5 pct of the ash and 68.3 pct of the sulphur were eliminated by four cleanings, that is, one cleaning and three recleanings of the concentrates. The remarkable point in the flotation of bituminous coal slurries is the high percentage of low ash coal available at 80 pct recovery.

With the types of slurries encountered, it was practically impossible to make a clean separation on one pass flotation and retreatment is the most important part of the problem. Different coals and types of feed require various degrees of retreatment, the same as most other cleaning processes and this is particularly important in respect to the reduction of sulphur. Increased floating time results in a better grade of concentrates and tailings. In laying out a plant, it is very important to make it sufficiently flexible so that retreatment may be varied to suit the particular condition.

The height of the froth column may be varied within rather wide limits (2 to 6 in.) without appreciably affecting the grade of concentrates or tailings. Increased depth of froth results in less capacity, greater reagent consumption, and low ash tailings. In cleaning up the tailings, both in the mill circuit and laboratory cell, the froth column must be of less height than while removing the major portion of the concentrate. If the same depth of froth were carried when cleaning up the tails as when removing the bulk of the concentrates, then more aeration and increased reagent consumption would be necessary, both of which would result in a less efficient separation.

REAGENTS

A number of frothing and collecting reagents have been used and, while observations made may not be true for all circumstances, they represent the conclusions reached under the existing conditions. Cresylic acid was found to be one of the best frothing reagents in respect to cleaning performance and together with kerosene and spray oil produced the best results in the mill operation. Extensive laboratory tests were carried out using pine oil, water-gas tar, and various prepared flotation oils, but cresylic acid was preferred for frothing. It was extremely desirable at Champion to obtain a froth that broke down readily, one that was not too tough, and one that was high in solids. Reagents were adjusted to produce the most desirable type of froth that could be obtained. The principal effect of kerosene appeared to be that of deadening the froth and helping to break it down. The collecting agents generally used in the mill circuit have been a No. 2 fuel oil or a mixture of 100 or 200 viscosity coal spray oil with kerosene used in the proportion of three parts of oil to one part of kerosene. Synthetic alcohols (frother) produce a more brittle froth with less reagent than cresylic acid, and the froth is of more bulk (higher in moisture) but easier to handle and pump. Because of its higher cost per gallon it was more expensive per ton than cresylic acid. In order to reduce the reagent cost, a tar acid oil produced in a low-temperature carbonization process was substituted for cresylic acid at a higher consumption but at substantial savings and equivalent cleaning performance to that obtained with cresylic acid. Step-oiling was used on several occasions and was found to be beneficial in respect to separation results but usually required longer treatment time, thus lowering the capacity and requiring more reagents. When recleaning primary concentrates no advantage was found in step-oiling.

It has been pointed out that improved separations are made at lower pulp densities, preferably at 10 to 12 pct solids in the cell, but at this lower solids there is an increase in reagent consumption over that required when operating at 20 pct solids.

ALKALINITY OF CIRCUIT

The entire water circuit of the washer is maintained alkaline by the use of lime in order to prevent corrosion of steel, and flotation was necessarily carried out in an alkaline circuit. It had been found in the laboratory investigations that the best results with respect to ash were obtained with a slightly acid circuit. Alkaline circuits have a tendency toward flocculation of the coal particles and therefore flotation results are somewhat impaired with respect to ash. For sulphur elimination best results are obtained by operating in a slightly alkaline circuit. Because the use of fresh water for dilution would increase the total water in circulation beyond a desirable point for best overall plant operation, either the filtrate from the concentrate filter or thickener overflow was used to dilute the primary feed and the primary concentrates. Due to the presence of colloidal clay, starch, and lime in these dilution waters, there is more interference in the separation than if fresh water were used. An excess of lime produces some oxidizing effect on the coal surfaces and inhibits the floating of coal. It is probable that a certain amount of lime is favorable (pH of 8.0) for the flotation circuit due to its electrolytic effect on the settling of clay.

EFFECT OF SLIMES AND COLLOIDAL CLAY ON FLOTATION

High content of colloidal clay in a flotation feed or in the dilution water inhibits flotation of coal particles probably because the surfaces are coated with a gangue or clay coating. Slimes and colloids appear to be more responsible for poor results than soluble salts. Prior to World War II, the flotation experience of Pittsburgh Coal Co. had been with slurry from deep-mined coal. During the war and afterwards, a considerable tonnage of strip-mined coal was processed daily in addition to deep-mine coal, with the result that a closed circuit could no longer be maintained in the washer; the solids content of the water increased greatly and caused coarser solids over 48-mesh to settle out in the thickeners. It was also necessary to add large quantities of lime and causticized starch to flocculate the clay. The combination of the presence of high percentage of solids in the water, colloidal clay, the high alkalinity by lime, the use of causticized starch for flocculation and the coarser particles in flotation feed made it impossible to operate flotation under the previous experience and practice. Because of war conditions it was impossible to undertake an investigation that may have developed a procedure for operating under the conditions brought about by cleaning strip coal.

PYRITE DEPRESSION

This subject was rather completely investigated in tests in the laboratory flotation cell and it is hoped that the results of this investigation will be the subject of a paper on this phase of coal flotation. Results on pyrite reduction on a mill scale are very inconsistent and generally show the same trend as the sulphur content of the raw coal. Sulphur elimination in an acid circuit is practically impossible and best results for both sulphur and ash elimination at Champion were found to be at a pH of 8.0 to 8.5. If necessary, a part of the alkalinity may be removed by sulphuric acid. The use of lime at the rate of 2 lb per ton of feed was tried in order to depress pyrite but no improvement in sulphur was noted (Tables 2 and 3, tests No. 8 and 9). It is believed that the conditioning time would be an important factor when lime is used as a pyrite depressant, but no conclusive evidence can be presented. The use of ferrous sulphate and ferric sulphate had been suggested for aiding in pyrite depression, but tests No. 10 and 12 (Tables 2 and 3) in which 0.50 lb per ton of ferrous sulphate and 0.48 lb per ton of ferric sulphate, respectively, were used, did not decrease the sulphur content. It appears that mechanical entrapment is one reason for pyrite's floating but some tests have indicated that the more dilute the pulp, the greater the floating of pyrite in the froth. Yancey[2] reports that the oxidation products of pyrite exert a stronger influence on the depression of pyrite than any action of a chemical reagent added. Therefore, the pyrite in a coal slurry should be easier to depress than a freshly prepared raw coal.

It has been the experience of the Pittsburgh Coal Co. operating flotation on Pittsburgh seam slurries that the reduction of sulphur is usually between 10 and 20 pct from the sulphur in the feed. Retreatment of primary concentrates is a more positive way to obtain the greatest reduction of sulphur than has been obtained by various depressing agents. Sulphur reduction may also be accomplished by pulling some bone into tailings at the expense of recovery.

HANDLING AND DEWATERING OF CONCENTRATES

The main practical problem in the flotation plant at Champion was that of handling the froth concentrate rather than in the separation itself. Settling tests had shown that it would require a large-diameter thickener to satisfactorily settle and thicken the concentrate; and, furthermore, there was no available area for a thickener in the immediate vicinity of the plant. All the flotation work in the mill-sized units was influenced by the necessity for obtaining a froth that could be satisfactorily filtered without thickening. The production of a froth that would break down readily was required and even some sacrifice in technical efficiency would have been made to obtain a more brittle and easily broken froth. The addition of chemicals for breaking down of the froth was tried, but excessive amounts were required in order to accomplish much in this direction. Without breaking down the froth, the two 8 by 14-ft drum-filter feed tanks were frequently overflowing; this overflow was returned to the flotation feed, causing an overload to the cells and creating an unsatisfactory operating condition for both flotation cells and filters. The best solution for breaking down the froth was found to be a bucket elevator operating at approximately 400 fpm. The original purpose of the elevator was to break down the froth by impact against the plate as the froth discharged. Actually the froth was mostly broken down in the buckets by the air pressure developed by the elevator before the froth reached the discharge point. By the use of the bucket elevator, the filters were operated without overflowing at a pulp density of 30 to 35 pct solids. When filtering the froth concentrate without breaking down the froth, the production of filter cake was at the rate of only 20 to 25 lb per sq ft per hr, but by breaking down the froth, this was doubled even with the dilute feed to the filters.

TAILINGS DISPOSAL

Tailings from the flotation cells enter No. 3 thickener and causticized starch is added to this feed at the rate of 0.5 lb per ton of solids plus 100 lb of lime per 7-hr shift. These reagents aid clarification and subsequent filtration of the underflow; after filtering, the cake is discharged to refuse.

CAPACITY OF SUBAERATION FLOTATION CELLS

The flotation cell used at Champion is the mechanically agitated subaeration machine and is the same in principle to the ore-separation machine, but has a double spitzkasten and froth discharge on two opposite sides of the cell. One of the fundamental differences between ore and coal flotation is in the tremendous bulk of coal concentrates both in respect to volume to weight, but also in relation to the concentrates produced per ton of feed. The coal cell has two paddle blades to remove froth instead of the single-blade paddles on the ore machines. The cells in use at the Champion plant are the 43- by 43-in. double froth discharge with a cubic volume of 50 cu ft per cell. The tonnage that may be handled by these flotation cells depends largely on the amount of minus 200-mesh in the feed. With a feed containing from 25 to 30 pct minus 200-mesh, the six-cell bank will effectively clean about 10 tons per hr, or an average of 1⅔ tons per hr per cell for the circuit consisting of four rougher and two cleaner cells. Therefore, rougher cells are handling over 2 tons per hr per cell. When the minus 200-mesh in the feed increases to 75 pct or more the six-cell bank will efficiently clean but 5 to 6 tons per hr, or slightly under one ton per hour per cell. Again, the rougher cells will handle more tonnage per cell, or approximately 1.5 tons per hr per cell. These capacities are cited as representative of the type of feed and conditions of operation at Champion. Putting the secondary tailings or middlings back into the primary cells did not cause any appreciable lowering of capacity in the primary cells and greatly improved the separation by making a lower ash concentrate at a high recovery. In coal flotation, it may frequently be advisable to sacrifice a certain amount of metallurgical efficiency in order to obtain a large capacity per square foot of cell area.

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[2] H. F. Yancey and J. A. Taylor: Froth Flotation of Coal, Sulphur and Ash Reduction. U.S. Bur. Mines *R.I.* 3263 (Jan. 1935).

A large flotation cell of 100 cu ft capacity has been developed for coal flotation and has a greater capacity per cell. The flared sides of the cell start closer to the bottom of the cell thus enlarging the surface over which froth is produced. This new cell design extends the activity created by the impeller in an outward direction and the agitation is reduced at the point of removing the froth. In the cell used at Champion, the original middling-return pipes were inadequate for returning the froth of four rougher cells through the middlings return of No. 1 and 2 cells. By increasing the diameter of these return pipes from 6 to 8 in., most of this trouble was eliminated.

With the new coal cell, the tailing compartment (including froth baffle and tailing gate) and the feed compartment are placed outside the cell. By this arrangement, it is easy to remove the middlings from any cell and transfer it to another cell by simple pipe connections.

Power

Three-phase 60-cycle power is supplied at 440 volts to the various motors in the plant. The power required to operate the various units connected with flotation is shown in Table 7.

TABLE 7—*Power Required by Various Units of Flotation Plant, 20 Tons per Hour Feed*

	Kw	
Feed Pump, from No. 1 thickener to Conditioning Tank		20.9
Flotation plant:		
Conditioning tank recirculating pump	5.3	
Reagent feeders	1.1	
12 cells flotation	36.0	
Froth elevator	3.4	45.8
Concentrate filters:		
2—8 ft by 14 ft filters	3.0	
2—Vacuum pumps	52.2	55.2
Tailings disposal from No. 3 thickener (35-ft diam):		
No. 3 thickener rakes drive	1.1	
No. 3 thickener underflow pump (2 in.)	5.2	
11 ft by 16 ft vacuum filter	2.2	
Vacuum pump for filter	23.9	
Filter-cake belt to main refuse conveyor	2.2	34.6
Total		156.5

Cost of Operating Flotation

Current costs are not available, since the plant has not been operated steadily for the past three years. In 1942, however, 42,282 tons of flotation concentrates were produced and direct costs are given in Table 8 for the operation, maintenance supplies, and power for the conditoning, flotation cells, and froth elevator. Pumping of underflow, operation of clean coal and waste filters, and so forth, are a part of the regular plant flowsheet, and therefore the cost of filtering concentrates, thickening and filtering of tailings, are not additional cost items when flotation is in operation. For this reason, the cost of flotation at Champion as shown by Table 8 does not include these items. Depreciation charges and conversion charge for a lower recovery due to cleaning also are excluded from these costs. Table 8 includes only the direct costs of operating the flotation units, conditioner, and froth elevator in 1942; reagent cost is included under supplies.

TABLE 8—*Direct Cost of Operating Flotation Plant at Champion* (*Based on Year* 1942) 20 *Tons per Hour Feed* 85 *Per Cent Recovery of Concentrates*

ITEM	DOLLARS PER TON CONCENTRATE
Labor:	
Operating	$0.041
Maintenance	0.009
Supplies, including reagents	0.072
Power (2.7 kw-hr per ton)	0.027
	$0.149

Reagent consumption has been shown in Table 3 and the cost of reagents may be readily calculated by applying the appropriate delivered price of reagents to the consumption. During the period of 1936 to 1944, the reagent cost was from 4 to 5 cents per ton of concentrate when operating on a 20 ton per hr feed of No. 1 thickener underflow and from 6 to 7 cents per ton of concentrates when operating on a 11 to 12 ton per hr feed from No. 3 thickener underflow. These figures would be somewhat higher under 1947 prices.

SUMMARY

The froth-flotation plant at Champion operated as a part of the regular cleaning circuit from 1935 to 1944. Its operation was temporarily discontinued in 1944 because of the introduction of strip mined coal containing a large amount of clay which made it extremely difficult to clear the water system and to float coal in the flotation cells. The flotation units were operated in the regular closed water circuit of the cleaning plant. Flotation concentrates were filtered by vacuum filters and subsequently heat-dried as a part of the ⅜ in. by 0 cleaned coal. The tailings were disposed to a thickener for settling of waste solids (such as fine clay) and subsequently filtered on a vacuum filter and discharged to waste as a filter cake.

Flotation is best adapted to the cleaning of minus 48-mesh coal slurries and better results are obtained by eliminating the plus 48-mesh from the feed. Recleaning of primary concentrates was found advisable to produce a low-ash concentrate and a high-ash tailings. A multicell circuit consisting of four rougher or primary cells and two cleaner or secondary cells was found to be well suited for the conditions and objectives at Champion. The pulp density of the feed had to be regulated to permit proper filtering and sometimes it was necessary to sacrifice cleaning efficiency to obtain satisfactory dewatering and filtering of concentrates. A high-speed froth elevator was found to be a satisfactory means for breaking down the concentrate froth to a point at which filtering of concentrates without thickening could be readily accomplished.

In conclusion, it might be stated that the technical problems of cleaning thickened slurries from deep-mined coal by froth flotation are not too difficult in respect to ash reduction. From an operational standpoint, the handling of concentrates and tailings presents more difficulties than the actual separation. The problem of cleaning slurries extremely high in colloidal clay such as strip-mined coal was not undertaken because it was not expected that the plant would operate with an appreciable tonnage of strip-mined coal for more than a few years.

ACKNOWLEDGMENT

The writer is indebted to Mr. J. B. Morrow, under whose direction this work was carried out, for permission to present the information contained in this paper.

The results accomplished at Champion and the data presented herein represent the combined work of many individuals on the staff of the Pittsburgh Coal Co.; in particular, the late Mr. T. J. Wardell, Messrs. S. B. Barley, J. T. Crawford, S. M. Parmley, C. P. Proctor, J. A. Younkins, and R. E. Zimmerman. The author gratefully acknowledges their help and that of others whose efforts made possible the presentation of this paper.

DISCUSSION

(*J. Griffen presiding*)

J. B. MORROW*—I am not going to talk about the technique because frankly I do not know what is going on. If I did talk about technique, somebody would ask me a question and I could not answer it.

I would like to point out, however, that there is a great difference in handling an aspirated coal or in handling settled coal with any type of thickener.

I think it was about 1928 when Tom Fraser and I handled coal up to ¼ in. and we got quite a kick out of it. We thought we were on the track. That was fine because that coal was screened as it came out of the mine. You all know that coal has an oily film on it. It is difficult to wet minus 28-mesh coal. After it has gone through the pumps and thickeners that film gets worn off and there remains an entirely different type of coal. There is a great difference in making tests on screened raw coal rather than after it has been settled out.

We have a long way to go before we ever arrive at the same techniques that are used in ore concentrations. The main reason for that is because we are not equipped to maintain a constant feed. By that, I do not mean a constant feed in tons per hr but I mean constant feed in size content. In other words, you do not want to vary from minus-200, let us say, up to 35-mesh or larger.

In order to get flotation to work and achieve the same result that has been achieved in metal work, we shall have to be a great deal more careful about getting constant feed conditions which we do not have today.

The main problem we had was in handling the froth. We dropped the froth 30 ft down into square bins and had the froth cascaded from one level to the next. In that way we tried to break it but that did not work. We put in 10-in. pumps to try to handle about a 2-in. flow and that did not work. Finally, we used the belt elevator.

As Mr. Davis told you, the idea was fine but it did not work in the way we expected; however, as long as it worked, we did not care too much.

I think, personally, that is our main contribution to the art of coal flotation—being able to handle the froth.

On the matter of handling sulphur reduction, we did some test work once in the laboratory. We took pyrite and it was finely ground, and then we took marcasite and had that finely ground. The efficiency of separation on pyrite was relatively easy. That was in an alkaline circuit, pH about 8, using sodium thiamide at that time. We did extremely well on pyrite.

On the marcasite, we did not do well at all and that caused us to have some checks made. We discovered that quite a fair amount of what we loosely called pyrite in coal is not pyrite at all—it is marcasite. Because of the difference in shape, it is much harder to handle.

When we started this work in 1928 we just tried this, that and the other thing, and got into a lot of trouble. Now, I see a crowd here listening to papers on the subject. With our friend, Mr. Lewis, increasing the price of coal fairly often, I think we are all going to realize that we cannot say $3 a ton if it costs us 75¢ in order to have it cleaned. Therefore, I look for the handling of fine coal by means of froth flotation to increase.

J. GRIFFEN—Mr. Younkins, you had quite a little to do with some of this early work.

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* Pittsburgh Coal Company, Pittsburgh, Pa.

J. A. Younkins*—I do not think I have anything to add to the discussion. Mr. Davis pretty well summed up the troubles and difficulties. I had some questions I wanted to ask the other authors, but I will defer that until the general discussion.

R. E. Zimmerman†—I believe this paper by D. H. Davis on the froth flotation of bituminous coal slurries is a valuable contribution to coal preparation literature and should be of considerable aid to anyone making a study of coal froth flotation or to anyone contemplating the cleaning of slurry or extreme fines by such methods.

Regarding the early research work done on coal flotation by the Pittsburgh Coal Co. at their various Champion plants, it was my experience and pleasure to have participated in much of the reagent investigational work and pilot plant operation of the various units described by Davis.

Although many of the problems incurred seem simple enough now, at the time we were first faced with them, they presented very serious obstacles indeed. That observation applies, of course, to most problems faced today by coal preparation engineers. The questions which appear difficult to answer now will become easy to solve in the future.

Where a need for removing the impurities from extreme coal fines exists and where the market condition is such as to demand and economically permit its use, coal froth flotation is an effective process for its accomplishment.

Davis describes the Champion flotation units as having run for several years, as part of the regular plant operation. There are also flotation units in operation in the anthracite field and numerous units abroad.

In addition to the fact that flotation processes will clean coal sizes that other known processes cannot touch, or will do only a partial job, it is interesting to note that, if necessary, extremely low ash concentrates may be obtained. Although not reported by Davis in his paper, ash as low as 2 pct was obtained from Pittsburgh seam coal, by recleaning the coal concentrates four or five times. The yield of such a product, of course, was not great, but it illustrates what could be done if someone really needed an almost ashless coal—and could afford to pay the processing cost. It is my opinion, however, that much work has yet to be done to make flotation of fine coal more attractive.

There is still a considerable field of investigation in determining cheaper and more effective reagents. The handling properties of the froth, as brought out by Davis, are of paramount importance. A brittle froth is desired that will break readily. I suspect there are newer chemicals on the market than those described which would have this property. Or in lieu of such a frother, perhaps an application of a vacuum chamber could be developed.

Regarding the cost of operation, Mr. Davis is careful to mention that his figure of $0.149 is merely the direct cost per ton. It is my opinion that, including necessary auxiliary equipment, actual cost would be in the neighborhood of $0.25 or more per ton of concentrates and that would not include drying costs.

I notice that Mr. Davis, in his paper, also remarked that flotation is only suitable for cleaning solids under minus 8-mesh. It is my belief that we could find other processes for cleaning down to, let us say, 100-mesh more cheaply, and then, take the minus-100 mesh, if it is economically desirable, to run in flotation units.

Our plan with the Hanna Coal Co. of Ohio is to clean 3⁄16 by 100-mesh on wet concentrating tables. We could take the minus 100-mesh and run that through flotation units if necessary. However as he pointed out the deleterious effect of clay, the minus 100-mesh in our coal out in Ohio, chiefly due to blending in strip coal, is extremely high in clay. So far, I doubt if it would be economically feasible to clean it by froth flotation. Our minus 100-mesh out there will run between 30 and 35 pct ash.

H. J. Gisler*—This paper stresses the importance of intense reagent conditions for flotation of coal coarser than 48-mesh. This, of course, bears out the necessity of a separate circuit for the coarse fraction where intense conditions can be employed without affecting the separation selectivity on the finer coal and ash fractions. Generally higher density and more intense cell agitation along with more reagents favor flotation of coarse coal when treated in the absence of fines. Under such conditions 8- to 10-mesh coal can be effectively floated.

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† Hanna Coal Co., St. Clairsville, O.

* Denver Equipment Co., Denver, Colo.

Does there appear to be any definite advantage in the double overflow froth feature on the mechanical flotation cells? Both the single overflow and double overflow type of machines are now in use in the Pennsylvania coal fields, apparently with equally satisfactory results.

J. GRIFFEN—I might make just one comment here. You can see that the flotation men think you ought to go cautiously. The mechanical gravity cleaning men think it is cheaper to go down finer. That is the type of argument that I think is beneficial because we will finally get the cheapest method.

Mr. Davis, do you have anything to say about the single and double overflow machine?

D. H. DAVIS (author's reply)—I would prefer the double overflow. After all, with bituminous coal, there is a voluminous froth that must be floated and it is our experience that a double overflow improves both results and capacity.

Because of the voluminous froth, we had to use four-paddle blades instead of two-bladed paddles for removing the froth.

O. R. LYONS*—I want to make some comments on Mr. Davis' paper. We ran some tests at Battelle on filter cake material that had approximately the same screen analysis as that which they were treating over at Champion.

We have checked his pulp densities pretty closely and we obtained better results at 10 pct, although we tried up to 20 pct and did a pretty fair job; however, it was not as consistent as it was around 10 pct.

We used an alkaline pulp, somewhere between 8 to 8½ pH, although we tried higher alkalinities, and we found that we could remove the ash and keep the ash content down quite consistently without trouble.

What we were trying to do, however, was to remove as much sulphur as we could, starting out with a sulphur analysis of 1.6 to 1.8 to a 2 pct maximum. We were able, by straight flotation on a material in a range of 28- to 35-mesh, top size, to get down to 1.25 pct sulphur.

We ran a lot of tests in which we classified the material and tried flotation on the classified products and we found that we got the best results by taking a minus-65-mesh material and floating it and taking the plus-65-mesh material and tabling it, ending up with an overall sulphur of about 1.22 pct.

J. GRIFFEN—Was that Pittsburgh seam coal?

O. R. LYONS—Pittsburgh seam.

J. GRIFFEN—From Pennsylvania or Ohio?

O. R. LYONS—From Pennsylvania. We also tried ferric and ferrous sulphates as depressors without any success. As far as we could tell, they had no effect whatsoever.

D. H. DAVIS—I neglected to say, during the presentation of my paper, that we obtained a definite reduction in sulphur by retreatment, rather than by just one flotation, and our best results on sulphur reduction were obtained by retreating the froth.

O. R. LYONS—Did you get much success with lime?

D. H. DAVIS—Yes—lime aided in sulphur reduction.

B. M. BIRD*—For the benefit of the others present, about what percentage of sulphur would you ordinarily have in your Champion coal? In other words, give those present some yardsticks for gauging what you were doing on the flotation.

D. H. DAVIS—Well, on the ⅜ × 0 it would run around 2 pct and on the raw minus 48-mesh about 2.5 pct.

B. M. BIRD—In other words, the flotation was doing relatively very well.

J. GRIFFEN—Of course, that is one thing that always interests me—you never use the same criterion. When it comes to gravity concentration, you set up a gravity and you analyze what floats at the gravity. That is your objective. When it comes to flotation, to run float and sink tests on such fine solids is quite a chore.

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* Battelle Memorial Institute, Columbus, O.

* The Jeffrey Manufacturing Co., Columbus, O.

With flotation you usually take laboratory work with a flotation cell as your criterion. Then you judge plant operation against ideal performance in the laboratory; am I not right, Mr. Davis?

D. H. Davis—In general, you can get results out of a plant just as you predict in the laboratory. The results of froth flotation in the plant, if anything, are equally as good as laboratory tests provided that the plant is being operated properly.

We made some test work and we tried cyanide in test work, but we did not carry out anything extensive in full-scale mill operation and I cannot state why this was not done. It did indicate some sulphur reduction. In the laboratory we made as many as a dozen re-cleanings of the froth. We kept working it down, and it is very interesting to learn how much low ash coal there is in the slurries that are pumped to waste.

J. Griffen—Gentlemen, I am not a flotation expert, but I jotted down a few notes and I think it may be worth while to summarize some of the points on which there seems to be a fairly general agreement.

In a flotation plant, you cannot get optimum results if you have too wide a size range. In other words, as Professor Mitchell said, minus-200 cannot be cleaned. He was thinking that you cannot clean minus-200 when you have a lot of coarser stuff; is that right?

D. R. Mitchell—That is right.

J. Griffen—These papers have indicated that if the feed is minus-200, there is one problem and acceptable results have been obtained if it is a deslime feed. You can handle much coarser feeds with satisfactory results.

I think Mr. Morrow put his finger on one of the most important things of all—that flotation is a tool. The best results are obtained only by integrating it with all the rest of the coal washing plant.

The probable reason we have failed commercially to get the best results is that we have not taken care of all of these variable feed conditions which, unless there were an interest in flotation, did not seem to be too important. Now we find out that they are very important. In a coal plant the run-of-mine coal may have only 2 or 3 pct minus 48-mesh in it when it starts into the plant, but, actually, if hour by hour information is available there are likely to be 6 pct at one time, 1½ pct at other times, and different figures at still other times. That does not mean much on run-of-mine coal tonnage but it means a very great change in the load conditions when flotation feed is considered in tons per hour that must be handled.

Then there is the problem of intermittent operation of a coal plant. There is not any question that flotation is going to do its best work if operated continuously.

I was very glad Mr. McCabe mentioned the vacuum tank which is being used on the continent for breaking down flotation froths. I am interested in that because of one thing. It is the usual practice in Western Europe to dedust their feeds to their coal washing plant—that is, at something like 28-mesh or 48-mesh—and if the coal is too wet, they actually may be dedusting and then follow that with washing out the balance of the slurry. Then they will float those products separately and it may be that because that is fresh coal, they are able to work with very much more brittle froths that are easier to break down in a vacuum tank. They do usually collect the slurry that is produced by the washing operation separately and often add it to the flotation plant that is handling what is gotten out of the raw coal, but it may be that with our system, the vacuum tank would not work so well. However, it is interesting to check. I believe in that report on the coal preparation practice in Western Europe which came out recently that machine is described and some information as to the increase in filter capacity resulting from the use of this vacuum tank is included.

Flotation of Bituminous Coal

By R. E. Zimmerman,* Member AIME

(New York Meeting, February 1948)

The separation of fine sizes of coal from its impurities by means of various flotation methods has become of increasing importance in the coal industry. This, no doubt, will be even more so in the future as the quantity of high grade coal is depleted, particularly that used for metallurgical purposes.

The purpose of this paper is to describe and report the results of research work done under the auspices of the Pittsburgh Coal Co. to determine the amenability of their bituminous fine sizes of coal to flotation, to investigate the value of various reagents and factors affecting their use, and compare the results between froth flotation and bulk oil flotation.

Compared with the preferential flotation of complex minerals, or ores, the flotation of coal is comparatively easy. However, this statement should not lull one into the feeling that there are no special problems involved. For instance, the separation of coal from pyrite in high sulphur coals is still far from being 100 pct successful. Size consist has considerable effect on reagent consumption and recovery. Other factors will be described and finally it must be remembered that in the flotation of coal a low priced material is being treated and thus economic considerations govern more closely its application than is generally the case in treating other minerals.

As a result of the research work done by the Pittsburgh Coal Co. on the flotation of coal, it was concluded that it was practical to clean the −48-mesh coal commercially by froth flotation and after several different types of flotation units and circuits were investigated by means of pilot plants, large scale units were installed at their Champion Nos. 1, 4 and 5 Preparation Plants for the cleaning of −48-mesh slurries.

These large scale operations have been recently described by D. H. Davis.[1] C. P. Proctor and C. B. Carlson[2] in their chapter on Froth Flotation of Coal in the AIME Seeley W. Mudd series book on Coal Preparation describe some of the practical applications of our Pittsburgh Coal Co. laboratory work in actual plant practice as well as summarizing part of the investigational work. The author presents this paper, however, as a supplement to their reports, and records the basic research accomplished in the laboratory investigational work as well as the results of experiments using bulk oil flotation. In presenting this work the author wishes to state much more work must be done in applying the principles of flotation to cleaning coal, especially in view of recent studies and practices in other mineral beneficiation fields.

Methods of Conducting Laboratory Tests

The bulk of the research work was conducted in the laboratory of the Champion No. 1 Preparation Plant. This plant has elaborate facilities for sampling and analyzing coal and proved to be an admirable location for the investigational work.

Four types of laboratory flotation ma-

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* Preparation Engineer, Hanna Coal Co., St. Clairsville, Ohio.

[1] References are at the end of the paper.

chines were used. There was installed a Denver Equipment Sub-A (Fahrenwald) mechanical agitation, 2000 gm batch machine. There was also a Denver Equipment continuous feed, 5-in., 6 cell machine of the same mechanical agitation principle. In addition, there was a General Engineering Co., MacIntosh pneumatic 2 cell, 2000 gm batch machine. Later, there was developed a special churn or agitator used in bulk oil experiments.

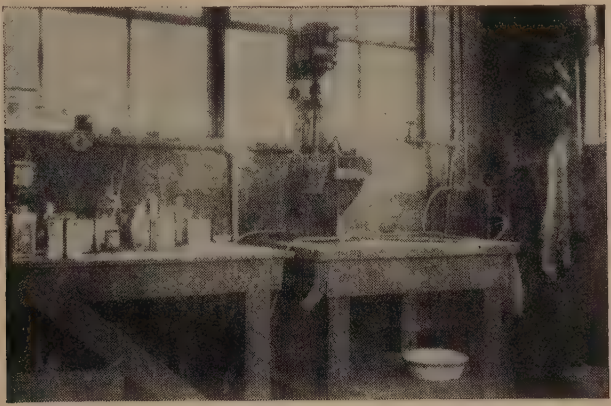

FIG 1—MECHANICAL AGITATION FLOTATION MACHINE.

Because of the difficulties in arranging constant and accurate feed rates for the 5-in., continuous 6 cell machine at the small capacities needed, it was found unsuitable to use this for research work. Similarly, although some excellent results were obtained from the MacIntosh pneumatic unit, it was found that for test purposes the single cell mechanical agitation machine was the best for speed and accuracy as it permitted close and measurable control. This machine is shown in Fig 1. Although generally listed as a 2000 gm capacity unit, its capacity was set at 1000 gm for coal because of the lighter specific gravity and the floating of the bulk of the feed. The MacIntosh pneumatic machine, as hooked up in the Champion laboratory, is shown in Fig 2.

The special churn used in some of the bulk-oil flotation work consisted of a steel, rectangular shaped box capable of holding, when half full (normal operating level), 5 gal of pulp. This box was supported at diagonal corners by shafting which was rotated at variable speeds, usually in the neighborhood of 24 rpm, and thus provided a back and forward motion to the pulp with a churning effect.

In setting up any series of tests it was essential to make sure: representative samples were used; that there was proper pulp density control; uniform or known water characteristics as to its pH value or soluble salts content; set time of floating, standard impeller speed, air suction, pulp level, rate of froth draw-off, quantity of reagents and uniform step oiling.

In measuring the effectiveness of a reagent, it was found that numerous factors have to be considered in determining its value, other than straight analytical data of the products. By that is meant, one must take into consideration the texture of the froth, its copiousness and consistency, and its brittleness or degree of toughness, the latter being of great importance in the handling of coal froth.

No attempt was made to measure the effectiveness of a reagent by the captive-bubble or cylinder flotation methods described by I. W. Wark[3] and in many other recent papers, including that by J. Rogers,

K. L. Sutherland, et al.[4] It was felt that actual tests in a flotation machine were of more practical value, as it is known that captive-bubble or cylinder tests are not necessarily identical with those obtained by a flotation machine. Regarding laboratory flotation tests as a whole it has been found that the individual operating the machine must use considerable care in following set procedures and have them clearly defined if they are to be duplicated by another operator.

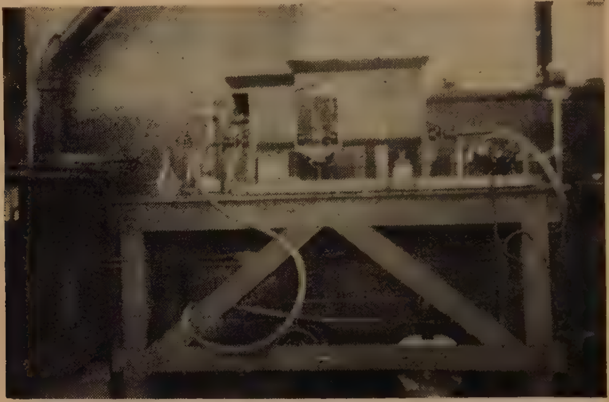

FIG 2—PNEUMATIC FLOTATION LABORATORY MACHINE.

FLOTATION THEORY AND DEFINITIONS

A great deal has been said and much has been written concerning the theory of flotation. It is not the author's purpose to go into detail concerning these various theories. Professor A. F. Taggart[5] states that, "Flotation is the art of wet concentration of minerals by floating the mineral to or near the top of the liquid and sinking the gangue or rock to the bottom."

In earlier years it was believed that only sulphide or metallic minerals could be separated by flotation. Later it was discovered that coal is readily floated or separated from its impurities. In recent years, however, it has been shown and is now widely practiced, that any mineral, metallic or nonmetallic may be floated and preferentially separated one from the other. This selective flotation has revolutionized mineral beneficiation.

Floatability is primarily a surface phenomenon. The nature of the film on the outside of a particle governs the degree or possibility of floating. Although some minerals naturally have this coating, others have it placed on them artificially by the use of certain chemicals. Conversely, chemicals may be added to remove this film and in a sense "gangue" can be floated and "concentrates" sunk by the selective use of reagents. Although not practical, as yet, it is feasible, for instance, to sink the coal and float the refuse by proper use of depressants and cationic reagents.

It should be clearly stated that the terms "float" and "sink" do not imply the usual meaning in coal beneficiation of floating a lighter gravity material and sinking a heavier material by means of their relative differences in specific gravity. Actually, in many mineral separations, the heavier particles are floated and the lighter ones sunk.

In froth flotation, bubbles of gas or air are generated by mechanical agitation or pneumatic means. The minerals to be recovered attach themselves to the bubbles and are carried upward to the top where they are skimmed off as "concentrates."

The refuse or "tailings" sink to the bottom. To aid in the formation of bubbles, to form a film around the bubbles and to strengthen them to perform their duty of carrying upward the desired minerals, certain reagents called "frothers" are added. Examples of such agents are cresylic acid, pine oil and various alcohols of the branch chain type. These agents must lower the surface tension of the water and form a film of oil around the bubble to permit the mineral to adhere to it.

To assist in the adhering of the mineral to the bubbles, it is usually necessary to add "collector" reagents.[6] In the cases of sulphide ores and coal, only a minimum of collector is needed. This is true where only one mineral is to be separated. However, when in selective flotation one mineral is first depressed in preference to another, as in the flotation of copper-lead-zinc, strong collectors are needed for bringing up minerals previously depressed. A collector coats the surface of a mineral so that it is not wetted by water but is attracted to and wetted by the oil film of the bubbles. Freshly ground non-oxidized sulphide metals and coal have this nonwetting characteristic naturally. Oxide and silicate minerals such as quartz, clays, fluorspar, and the like, are readily wetted by water and therefore resist the oil films surrounding the air bubbles.

It is possible by the proper use of collectors to change the coating of any mineral in such a way as to make it water repellent and, therefore, floatable. Examples of collectors for the sulphides and native metals are the xanthates, dio-thio-phosphates, thio-carbamates. Collectors for non-sulphide minerals are oleic and other fatty acids and soaps, fatty alcohol sulphates and mineral and coal tar oils. Coal is aided by both of the above groups.

For oxides and silicates the collectors are the so-called "cationic reagents" examples of which are the amines such as di-n-butyl amine or the longer chain compounds such as lauryl or stearyl amine hydrochlorides. Much of the recent flotation literature concerns cationic reagents. For example, J. D. Williams[7] describes the flotation of cement rock by means of lauryl amine and Donald Scott et al[8] describe the amine flotation of gangue from magnetite concentrates.

In addition to the above two major groups of reagents classed as "frothers" or "collectors," other classes of chemicals are frequently employed. For example, "depressors" are used where one mineral is to be prevented from being carried up by the bubbles in preference to another. These agents coat the particles to be depressed so that they will be readily wetted by the water and not the oil film surrounding the bubbles. Such depressors are the cyanides, lime and oxidizing agents as the chromate, and ferrous and ferric salts.

Sometimes "deflocculating" agents such as caustic soda or sodium silicate are needed to deflocculate slime and clean mineral surfaces. "Activator" or "promotor" agents may be used as when copper sulphate is added to zinc sulphide flotation. The alkalinity of the solution is usually very important and a knowledge of the most effective pH value must be known. Sometimes an acid circuit is much more desired than an alkaline one.

It can be readily seen that froth flotation is far more complex than merely adding a few drops of oil to a pulp and agitating. It is becoming a highly technical and intricate art.

Before describing the results of test work at Champion No. 1, another type of flotation should be briefly described. That is, bulk oil flotation. In froth flotation, chemicals are added in small quantities—frequently less than 0.25 lb per ton of feed. In bulk oil flotation, large quantities of oil are added, from 5 to 50 pct of the feed, forming an amalgam of the mineral and the oil. This process is a mass process where the refinements mentioned previously could not apply but in dealing with coal, for instance, may have some value. Advantage is taken of the affinity of oils such as petroleum or coal tar oils towards coal, surrounding and wetting the coal particles and not wetting oxides and silicates such as clay, shale, sand, and the like. The amalgam formed is screened off from the water containing the impurities. It is especially effective in separating coal from clay and on extreme fines such as −200-mesh. It cannot, however, differentiate between boney coal and purer coal but will collect all highly carbonaceous material and also sulphides such as pyrite. This bulk oil process has been applied commercially as the Trent Process. It is necessary, of course, to carbonize or distill off the oil for re-use.

Flotation of Bituminous Coal

Most of the flotation experiments were conducted with coal coming from the Champion No. 1 Plant. This is Pittsburgh seam coal chiefly from Washington County and is a high volatile, high rank bituminous coal. It is strongly coking but too high in both inherent and extraneous sulphur to be used for metallurgical coke. Impurities tend to increase in the finer sizes.

The Champion Plant is equipped with the Rheolaveur system for washing coarse coal and 3/8 × 0 fine coal. This system is not generally effective for efficiently cleaning coal below 48-mesh size and in the course of operations the −48-mesh accumulates in Dorr Thickeners. This is the material which it was desired to clean by flotation and, therefore, it was largely used as the raw material for test work.

The size split at 48-mesh was not sharply drawn and varied under different plant operating conditions. A typical analysis and size consist under normal operating conditions is shown in Table 1. In general, this analysis would govern in most of the tests. In running any one series of experiments, however, care was taken to use part of the same sample of Dorr Thickener underflow or sludge.

Table 1 shows that the Dorr Thickener underflow is practically all through 28 mesh, with 11.7 pct + 48 mesh. Note that this material contains 33.3 pct −200-mesh running 27.3 pct ash. Head ash is 17.0 pct. The 100-mesh is relatively low in ash indicating that the Rheolaveur system at Champion does a fair job down to this size as far as ash reduction is concerned.

The head sulphur is 2.95 pct. This impurity likewise concentrates in the finer sizes reaching 4.50 pct in the −200 mesh. Regarding sulphur, it is interesting to note that this coal contains the high amount of inherent sulphur of 1.35 pct of both the organic and sulphate variety. The balance is in iron sulphide and although usually referred to as pyrite, is more of the marcasite variety of FeS_2.

Table 1—*Flotation Heads or Feed (D.U. Underflow Slurry)*
(Sieve Analyses are Tyler Standard)

Size	Pct	Cumulative Pct	Ash*	Sulphur*
Plus 28 mesh	0.5	0.5		
28 × 48	11.2	11.7	7.6	2.20
48 × 100	32.8	44.5	9.8	2.30
100 × 200	22.2	66.7	14.3	3.05
Minus 200	33.3	100.0	27.3	4.50
Head	100.0		17.0	2.95

* Ash and Sulphur analyses on "Dry Basis."

The ordinary float and sink method of determining the possible ash and sulphur reduction by floating in liquids of various gravities which is so suitable for coarser coals is not applicable to − 200 mesh. Table 2 is presented, however, to show the results of gravity separation on the +200 mesh. This indicates that at a specific

Table 2—*Float and Sink Analysis of Plus 200 Mesh Flotation Feed (D. U. Underflow)*

1.35 Float			1.35 × 1.55			1.55 Sink		
Wt. Pct	Ash	Sul.	Wt. Pct	Ash	Sul.	Wt. Pct	Ash	Sul.
64.3	4.7	1.75	24.1	14.6	2.80	11.6	45.0	6.55

gravity of 1.35 one can recover 64.3 pct of the coal which will analyze 4.7 pct ash and 1.75 pct sulphur. Combining the 1.35 float and the 1.35 × 1.55 float fractions, a recovery of 88.4 pct may be obtained at 7.4 pct ash.

To determine how much recoverable coal may be obtained by flotation and what one may expect in regards to yield, ash and sulphur reduction from the described Dorr Thickener slurry shown in Table 1, a series of tests were made in the mechanical agitation laboratory machine in which a rougher concentrate was first made giving a very high ash tailings. This rougher concentrate was then re-cleaned six times, each corresponding concentrate and tailing analyzed for ash, sulphur and weighed recovery.

The first five passes were made by froth flotation. The sixth or final cleanup was made by changing the system from froth flotation to a bulk oil or Trent treatment. This latter method was found exceptionally well adapted for removing the final ash impurities in the form of clay.

In the froth flotation phase, the following reagents were consumed:

Lime	3 lb per ton feed
Cresylic Acid	0.75 lb per ton feed
Kerosene	0.87 lb per ton feed

In the bulk oil or Trent treatment, 15 pct by weight of kerosene was added in making a final cleaning of the sixth pass. The method used in the Trent treatment will be described later. The lime added to the initial feed was added primarily to depress sulphur. Diluting water had a pH value of 7.2. With lime the pulp pH was raised to 8.5. As more diluting water was added in the later stages, this was gradually reduced by the last re-cleaning to 7.2. Half of the kerosene and cresylic acid was added in the first pass and the balance by step oiling throughout the runs in froth flotation. In bulk-oil phase, all of the 15 pct kerosene was added at one time.

The results of this re-cleaning test are shown in Fig 3. The cleanest concentrates obtained, after six passes, was 3.6 pct ash at a yield of 52 pct. 81.5 pct was recovered at 7.6 pct ash. Yield drops off rapidly after 6.0 pct ash is reached. From a practical full scale flotation operation viewpoint, it is believed that the economical range would be between 7–9 pct ash concentrates which would give a recovery of 80–88 pct. Note that this would require re-cleaning the first or rougher concentrates twice. Actually, with a multiple cell continuous machine this can readily be done, as well as re-cleaning of the tailings. Later, large scale plant operation was able to come within the desired ash-recovery range without any serious trouble.

The sulphur curve shown in Fig 3, is somewhat erratic. It was always difficult to check sulphur results. In this series of tests only lime was used as a depressor. Although a more detailed discussion of pyrite depression will be discussed later, it should be stated here that the conclusion reached on pyrite or marcasite depression was that although other depressors may be more effective, lime, because of its lower cost, was the most practical depressing agent. Further, it was soon learned that in spite of any depressing agent the greatest single factor interfering with removing pyritic material was that of mechanical entrapment in the froth itself due to the large bulk of coal being floated off; and that the best and most effective method was to re-clean the froth by one or more passes through the machine.

Fig 4 shows the recovery and ash in the concentrates according to size consist throughout the six re-cleaning operations previously described. As Table 1 shows, the raw 28 × 48-mesh increment is relatively clean running 7.6 pct ash, and other increments progressively dirtier reaching 27.3 pct ash in the −200 mesh.

The inherent ash in this coal is apparently slightly below 3 pct. Repeated tests regardless of reagents used, failed to reduce the ash any lower. In investigations of other Pittsburgh coals, the lowest ash concentrate ever obtained was 2.7 pct. Recovery, of course, drops off very rapidly in these final low ash stages. Note that the −200 mesh curve crosses the 100 × 200-mesh curve, giving a cleaner product in the final stages.

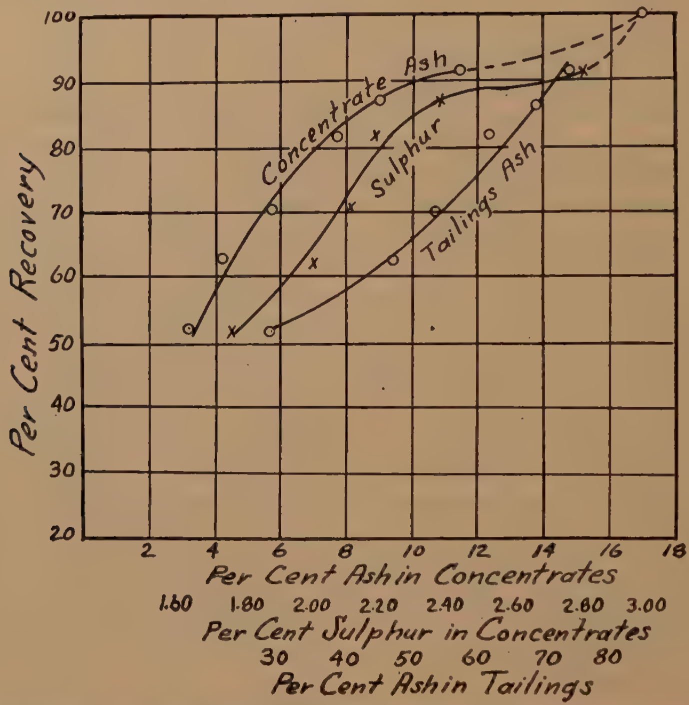

FIG 3—CONCENTRATE ASH AND SULPHUR AND TAILINGS ASH VS. RECOVERY—CHAMPION NO. 1 DORR THICKENER UNDERFLOW.

John T. Crawford[9] in his studies of bituminous coal flotation states that in the flotation of coal, the −200 mesh particles float off first, followed by the 100 × 200-mesh and then the coarser sizes in order of their increased particle size.

A. M. Gaudin et al,[10] in a study of the flotation of minerals such as galena and sphalerite, state that the medium coarse particles will float off first and that the extreme fines are sometimes very difficult to float and slower acting.

It has been the author's experience that this is often true in coal flotation. The coarser particles will float last and a large proportion of the −200 mesh will float first, but there will also be a proportion of −200 mesh which is harder to float and that is the extreme fines of perhaps under 20 microns. This material can be classed as slimes or colloids and an excess of these extreme fine sizes is detrimental to flotation, slowing up the reaction, causing excessive reagent consumption and poor recovery.

A. M. Gaudin et al state in their study two possible hypotheses which may explain the reasons for the inhibiting effect of extreme fines, to quote:

"3. If fine particles are sufficiently small to be in Brownian movement, the molecular bombardment may be sufficient to dislodge them from gas-bubble boundaries, thus preventing effective flotation.

“4. If the probability of fine particles adhering to air bubbles, or having air precipitated on their surface is substantially smaller than the similar probability in the case of coarser particles, fine particles should not float as completely as coarse particles and more time should be required to complete flotation.”

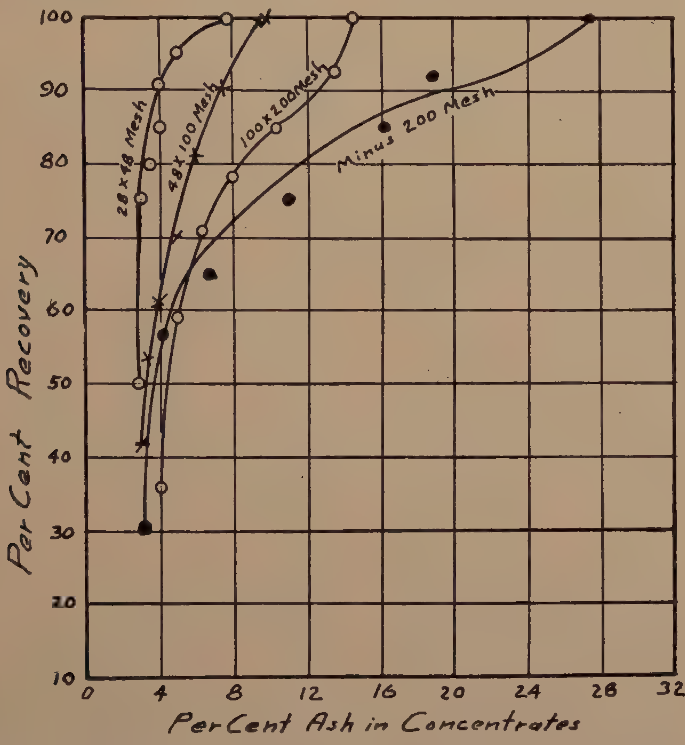

FIG 4—RECOVERY AND ASH IN CONCENTRATES ACCORDING TO SIZE CONSIST.

The largest sizes that can be floated in froth flotation depend upon various factors such as specific gravity, wetability using the most effective collector, shape of particle and the surface tension of the fluid. Generally speaking, it is possible to float larger coal particles than heavy metallic minerals. Particles as large as $^3/_{16}$ in. were floated in the laboratory machine at Champion No. 1. To float these coarse particles, however, the feed would have to be closely sized and it would be necessary to use such strong collectors and frothers that an impure concentrate would result.

It is far more economical to clean these coarser particles by some other means. Experience at Champion No. 1 was that 48-mesh was the most practical and efficient top size to clean by flotation. If excessive coarser sizes should enter the feed, it would be better to screen or classify them out of tailings rather than attempt to float them.

PULP DENSITY IN FROTH FLOTATION

As pointed out by John T. Crawford[9] pulp densities normally used in froth flotation of metallic ores are of little guide to desired densities in floating coal. This can be readily appreciated when one realizes that the specific gravity of ores may be over 3.0 while that of coal is about 1.5. Another factor is that in coal flotation the bulk of the material is floated while in metallic ores the percent of mineral floated is very small. A third factor is that the bulk of coal slurry treated, such as Champion No. 1 Dorr Thickener underflow is −200-mesh and these extreme fines require greater pulp dilution than coarser particles.

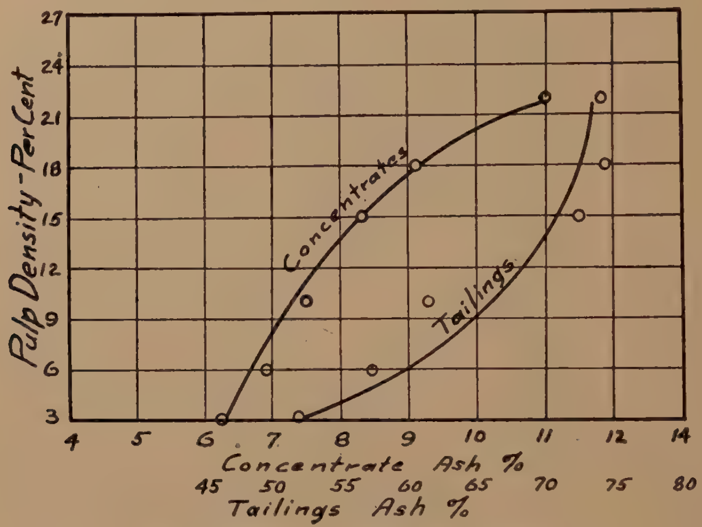

Fig 5—Effect of varying pulp density.

Metallic ores are usually floated with pulp densities ranging from 20 to 30 pct solids. If one were treating relatively coarse bituminous coal, say −28-mesh with a minimum of −200-mesh particles, good results could be obtained with a pulp density of 20 pct. In treating Champion No. 1 underflow of size consist shown in Table 1 best results were obtained with dilutions of 10–12 pct solids.

Higher densities result in higher ash concentrates. Lower densities result in cleaner concentrates but also somewhat lower ash tailings. Extremely low densities cause excessive reagent consumption and an excessive use of water. The effect of pulp dilution in floating Thickener underflow is shown graphically in Fig 5. The same amount of reagents was used in all tests, namely, 2 lb CaO; 0.5 lb Cresylic Acid; and 0.65 lb Kerosene per ton of dry feed.

Actually in a continuous flotation mill circuit the pulp dilution varies at different points in the circuit. Normal thickener underflow will run 50 pct solids. At this consistency lime and part of the collecting reagent are usually added in the pump or conditioning tank. Diluting water is added at the feed to the first cell to 15 to 20 pct solids. Each succeeding cell will then have a more dilute pulp as concentrates are removed. Where concentrates are recleaned water is added to keep the dilution down.

Water and Effect of pH Value

Generally speaking, the pH value and the amount of soluble salts in the water used in flotation are important factors. In the flotation of many minerals it is absolutely necessary to control these factors for successful separation. Collecting agents especially are most sensitive to pH as well as depressors or dispersing agents.

An example of the importance of proper pH control is ably illustrated in a paper by H. H. Kellogg and H. Vasquez-Rosas[11] in their research on the use of long chain primary amines as a collector for the flotation of galena-sphalerite ores. Maximum contact angles for each mineral were obtained within relatively narrow ranges of pH values

To date, incomplete data are available in the study of the full effect of pH value in coal flotation, especially with the use of complex chemical reagents and it is to be hoped that a more scientific study will be made of this phase of coal flotation in the near future.

In the investigation work at Champion No. 1 and in further tests at other plants of the Pittsburgh Coal Co., the effect of soluble salts in the water appeared to be rather negligible. A series of flotation tests using the same coal showed only a slight retardation with the use of water containing a high amount of soluble salts. The Champion No. 1 water supply had an alkalinity of 100 parts per million and a pH value of 7.5. The amount of soluble salts in fresh make up water to the plant contained only 46 parts per million of sulphates and 24 parts per million of chlorides.

The alkalinity or acidity of the water, measured in terms of its pH value, affects both the flotability of coal and pyrite. On the acid side a better grade of concentrates is obtained than on the alkaline side. Unfortunately, this holds true of both coal and pyrite. Acid circuits were found to be better for dispersing of slimes and colloids. A higher recovery, however, was obtained in slightly alkaline circuits, that is, pH of 7.0 to 7.5. With increase of pH, recovery and grade dropped appreciably, particularly, after 8.5 pH. This is shown in Fig 6. Due to the advantages in depressing iron sulphide, however, it was concluded that a solution of 8.0 to 8.5 pH was the most desirable range in treating Champion No. 1 slurry and was usually obtained by the addition to the flotation circuit of 2 lb of lime per ton of coal treated.

In the tests shown in Fig 6, Champion No. 1 slurry of 17.0 pct head ash and 2.95 pct sulphur of similar analysis to that shown in Table 1 was used. Champion No. 1 fresh water was used for dilution and pH controlled by additions of lime or sulphuric acid. Analyses are the results of six separate tests at different pH values. Rougher or single pass concentrates only were collected. A standard frother of 0.45 lb of cresylic acid and 0.65 lb of kerosene collector per ton of feed was consumed in each test, with five minutes conditioning time of pH regulator and fifteen minutes flotation time. The coal sample for each test consisted of 1,000 g.

FLOTATION REAGENTS

In the first part of this paper it was mentioned that flotation reagents were divided into various classifications according to their purpose. In general, they come under the major headings of frothers, collectors, depressors, and dispersing agents. There is an almost endless variety of compounds and, in a sense, there is no such thing as choosing permanent reagents because there are continuously being added to the market new chemicals which may be cheaper and more effective than previous ones. Further, a chemical used for treating one coal under certain conditions may not be as effective as others under different conditions. Each individual application must be studied separately.

During the course of the Pittsburgh Coal Co.'s investigation over one hundred different reagents were tested. Admittedly, many of them were subjected to only brief tests and it is conceivable that some which were rejected, would under a more close and thorough investigation prove to have considerable merit. The author feels that especially in view of the importance of finding the proper pH range for each individual reagent, a re-investigation of many of the chemicals should be made.

Coal is comparatively easy to float.

Freshly crushed coal without any reagents will partially float if dropped into fresh water, or a few drops of frothing agent alone may do a fair job. Where it is necessary, however, to make a more accurate separation with maximum recovery and at the same time to depress iron sulphides, bone coal and other deleterious material, the choice and use of proper reagents becomes more complicated.

Another factor of equal importance in coal flotation, and previously mentioned by D. H. Davis[1] is the consistency and toughness of the froth. A coal froth must be tough enough to have sufficient carrying capacity, and at the same time be sufficiently brittle to break down readily for subsequent handling and filtering.

Finally, coal, a relatively low priced mineral must be separated as cheaply as possible and therefore, cheaper though inferior reagents may be employed.

Many of the older coal tar products mentioned in coal flotation literature were found to be unsatisfactory due to their non-selectiveness and too tough a froth. Under this category are crude coke oven tars and creosote oils. Although many frothers have considerable "collecting" properties and vice versa, and could be employed by themselves, it was found necessary to employ both collectors and frothers.

Of frothers, the choice finally narrowed down to the use of pine oil or cresylic acid. It was found that steam distilled pine oil was the most powerful frother. Cresylic acid, however, produced a cleaner grade of concentrate and a more brittle froth. This, plus the factor of price, decided our choice of this agent for our frothing agent.

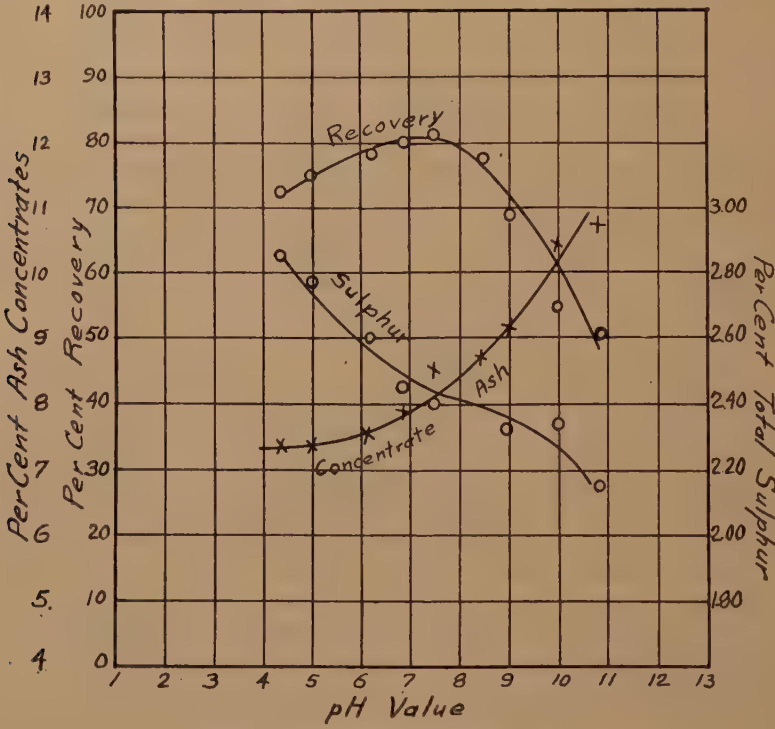

FIG 6—EFFECT OF VARYING pH VALUE.

Cresylic acid is a loosely applied term pertaining to alkyl phenols boiling above 190°C and may consist of mixtures of ortho-meta and para-cresol as well as xylenols and a variety of higher boiling homologues. It may be obtained from petroleum, high or low temperature tars. At Champion No. 1, because of the close proximity of the Disco Low Temperature Coke plant, use was made of the cresylic acid from this process and it was found just as effective as that obtained from other processes. The consumption of cresylic acid as a frothing agent varies with the numerous conditions mentioned in this report. Under so called "normal" plant conditions using Dorr thickener underflow slurry, where kerosene or fuel oil was used as a collector and pH raised to 8.0–8.5, usual consumption would be 0.45 to 0.65 lb cresylic acid per ton of feed.

A wide variety of collecting agents was tried out. They included such reagents as the xanthates, aerofloats, thiocarbanilid, oleic acid, tar oils, crude petroleum oil, kerosene, fuel oil and low temperature tar fractions. Although, as previously mentioned, a closer and more scientific control may readily bring out the value of many of the newer organic compounds, it was our experience that most of them were unsuitable and too strong for coal flotation, due to the highly carbonaceous character of the impurities, difficulty in removing pyrite and, quite importantly, the physical characteristics of the resultant froths.

The choice for collectors narrowed down to certain petroleum products, such as fuel oil, coal spray oil, kerosene and even straight Penna. crude petroleum. Water gas tar was found to be effective also. Of these mentioned, kerosene and fuel oil gave the most consistent and best results. In plant practice it became largely a matter of price and availability although laboratory tests showed somewhat better performance with kerosene.

These collectors produced a heavy mineralized froth concentrate, a saving in frother reagent consumption, gave a higher percent solids in the concentrates and at the same time were unusually selective in separating coal from its impurities. Kerosene, especially, appeared to have the property of flocculating and helping to break down the froth. In fact, if used excessively, it would tend to destroy the froth before the froth could be removed from the machine.

The consumption of either fuel oil or kerosene would normally amount to 0.45 to 0.65 lb per ton of feed, which, as above mentioned, is the same range specified for the use of cresylic acid.

Our concern with depressing agents was almost entirely for the purpose of depressing iron sulphides. Some work was done with the use of dispersing agents such as the silicates to disperse slimes, but innumerable tests were made with various depressors for the purpose of removing iron sulphides.

Unfortunately, although most of the coal flotation laboratory investigations were later duplicated in the plant installations with very close agreement, such was not the case when attempts were made to depress iron sulphides on a large scale. It was soon discovered that in dealing with large tonnages, the voluminous froth produced brought large quantities of iron sulphides with it in spite of any practical quantity of depressor which could be added.

This action was called mechanical entrapment and it was dealt with chiefly by re-cleaning or re-floating the concentrates once or twice more.

Aside from mechanical entrapment there was found to be a wide variation in the floatability of iron sulphides due to its character or structure. Much of the so-called pyrite is frequently in the form of marcasite. Although the only difference between marcasite and pyrite is one of crystal structure, the former being orthorombic and the latter isometric, it was found that marcasite was more difficult to depress than pyrite and that the iron sulphide in Champion No. 1 slurry was more of the marcasite variety.

The floatability of pyrite or marcasite also varied according to the amount of laminated particles containing layers of coal, the latter naturally being difficult to depress.

On the other hand, oxidized or partially oxidized iron sulphide particles were difficult to float and it is believed that the oxidation products of iron sulphide have a far more depressing effect on pyrite or marcasite than any outside depressing agent added, as can be demonstrated easily by anyone comparing freshly ground pyrite with that found circulating in coal washery water. The effect of natural oxidation products on pyrite is described by H. F. Yancey and J. A. Taylor.[12]

Certain types of compounds are more suitable for depressing pyrite and/or marcasite. They are usually of the divalent sulphur or trivalent nitrogen type. The groupings which should give a wetted surface are the hydroxyl, the sulphonic, the nitro, the carboxyl and any similar radical which makes the compound more soluble. Taggart[5] mentions the use of gelatine, starch, glue and other colloidal substances as being effective. The author's experience with the latter type of agents is that they will also depress coal with equal effect.

Some of the most effective reagents for depressing iron sulphide as far as laboratory testing is concerned are listed as follows: Sodium or potassium cyanide. Lime. Sodium carbonate or bi-carbonate. Pyrogallic acid. Para-aminophenol. Di-potassium phenol sulphonic acid. Sodium and potassium hydroxide. Permanganates. Ferrous and ferric sulphates. Sodium sulphite.

As previously discussed, pH control is essential in the effective use of any reagent. It is certainly true of depressing agents. H. F. Yancey and J. A. Taylor[12] point out that the iron sulphates, which they found to be the most effective depressant of pyrite were efficient between 4.6 and 6.9 pH but lost their effectiveness sharply on both sides of this range. Where lime is used as a depressant the higher the pH the greater the depressing effect. This is illustrated in Fig 6. Since high pH likewise depressed coal, a compromise was reached by operating at not over a pH of 8.5.

Sodium or potassium cyanide depresses pyrite, and tests using cyanide up to 0.3 lb per ton feed would bring Champion No. 1 slurry containing 2.95 pct sulphur down to 2.35 pct. Sodium cyanide used in combination with lime would further reduce sulphur content to 2.10 pct.

Under the same conditions, however, lime alone would reduce sulphur to 2.40 pct (rougher concentrates, with no recleaning) and since cyanides are expensive, there was no point for its use in practice. The effectiveness of cyanides also appear to be greatly reduced by mill water containing a high amount of soluble salts.

In almost every case, in the study of depressants, the cost of the reagents in respect to their final effectiveness in plant practice prohibits their use under present economic conditions. Conceivably, however, where good, low sulphur coking coals are becoming scarce their use should be considered in spite of the many limiting factors mentioned by the author.

Under the conditions at Champion No. 1, it was concluded that lime alone was sufficient for our needs. Lime is not only relatively cheap, but serves to keep the system alkaline, controls pH value and aids, in general, settling of solids in thickeners and settling tanks.

In the laboratory it was found that the rate of 2 lb of lime per ton of Dorr thickener slurry (dry basis) was sufficient to produce the desired 8.0–8.5 pH. In plant practice where the entire water system would be kept at that pH range the actual quantity would vary depending upon the many plant variables. An important factor in the use of lime is that it is desirable to have as long a conditioning or contact time with the coal-iron sulphide feed as possible and therefore, it would usually be added at the farthest point from the flotation machine.

The author wishes again to make clear in the final selection of reagents both as to kind and quantity, that an investigator or coal preparation plant operator must study his own peculiar conditions to determine what reagents he should use in his own particular problem.

Table 3 recapitulates not the best results ever achieved but the results obtained considering economic and other practical factors involved in setting up a commercial flotation installation at Champion No. 1. The figures are obtained from laboratory tests. Later, plant practice checked these results very closely except that sulphur content was usually 0.10 to 0.15 pct higher in the plant clean coal product. Note that it is considered that at least one re-cleaning of froth concentrates is necessary.

BULK OIL FLOTATION

Earlier in this paper a brief reference was made to experiments using bulk oil flotation as distinct from froth flotation, and to the fact that the bulk oil method may have value in situations where extreme fines (a high percentage of —200-mesh) exist, or where the coal washery system contains a high percentage of clay.

The need for bulk oil flotation in the research at Champion No. 1 arose when it was found difficult to remove last traces of clay from concentrates or where recovery was greatly affected by slurry feeds containing a great amount of slimes or colloidal matter. Having made some preliminary tests which showed promise, further work developed some interesting facts not heretofore believed known.

Bulk oil flotation is where a large quantity of oil, say 5 to 50 pct, (usually a crude petroleum or coal tar oil), is agitated in a water solution containing sulphide minerals, or carbonaceous or bituminous particles and their impurities. Due to the natural affinity of oils toward these kinds of minerals, an oily mass called an amalgam is formed containing coal or other minerals. The impurities in the form of clay or other oxides are left in suspension in the water solution. When used in connection with the cleaning of coal it is called the Trent Process.

Obviously, this is not a highly selective process in cleaning a coal which contains bone coal or carbonaceous laminated particles, because the large quantity of oil used will collect this material with the coal. It will also collect iron sulphides with the coal. An almost perfect separation or division occurs, however, in dealing with clayey matter and other non-sulphide and non-carbonaceous material.

In an attempt to find the lowest ash obtainable from Champion No. 1 Dorr Thickener underflow slurry, the final clean-

TABLE 3—*Reagents, Recovery, Ash and Sulphur Reduction, by Two Pass Froth Flotation on Champion No. 1, Dorr Underflow*

Reagents Used per Ton Feed	Condition Time,* Minutes	Float Time, Minutes	Product	Pct Ash	Pct Sul.	Pct Recovery
2 lb. CaO	5		Heads	17.0	2.95	100.0
0.55 lb Cresylic Acid	1	15	Rougher conc.	9.0	2.50	89.5
0.65 lb Kerosene	2	8	Re-clean conc.	8.2	2.25	80.0
pH Value 8.5			Comb. tailings	66.3	5.58	20.0

* Treatment of 1000 g sample (dry basis). 0.35 lb Cresylic Acid and 0.45 lb kerosene used in first addition, balance added by "step-oiling" during part of first and second pass. All reagents calculated on a per ton of dry raw coal basis.

ing of froth concentrates after five passes still showed evidence of clay. A sixth pass or re-cleaning was made by bulk oil flotation reducing the ash from 4.5 pct in the fifth or last froth concentrate to 3.6 pct. The final low ash concentrates shown in Fig 3 and 4 were obtained in this manner.

The normal bulk oil or Trent concentrates, called amalgam, collect into an oily mass which, with cessation of agitation, will generally settle to the bottom of the machine. To separate the tailings it is necessary to drain the contents out onto a screen, the amalgam being retained on the screen and the water and impurities draining through.

First test work was accomplished in the same single-cell batch, mechanical agitation machine employed for froth flotation, using 1000 g samples, adding a given percentage of oil, and agitating generally for about 8 min., or until the amalgam has become well formed. Although working well in dilute solutions, good separations can be made at higher pulp densities up to 30 pct solids. If the higher densities are used, it will usually be necessary to wash the amalgam with fresh water to remove traces of impurities entrapped in the mass.

Higher percentages of oil will speed up the reaction as well as give a lower moisture product. The kind of oil used also has a great deal to do with the speed of formation and the physical characteristics of the amalgam.

As in froth flotation, a wide variety of agents was tested such as petroleum oils, coal tars and coal tar oils, vegetable and animal oils. All these formed amalgams and with very little difference in ash content. The physical characteristics, however, varied as well as the quantity of agent required and the length of time required to agitate. No effective depressant of iron sulphide was found and pH value appeared to have little significance. Some reduction in sulphur was obtained, however, by reagitation and thorough washing of amalgam. Of coal tars, water gas tar gave superior results. Of petroleum products, crude oil, fuel oil, or kerosene was the best. Motor gasoline and benzol were also effective.

One objection to the ordinary amalgam is that of difficulty in drawing the moisture from the irregular shaped and frequently sticky mass. It was found that continued agitation in the machine, especially at lower speeds, produced an entirely different type of product. The amalgam would separate into small fish-egg shaped particles called "caviar." This type of product drained water much more readily; and whereas moisture might be as high as 25 pct for amalgam, when "caviar" was formed it would be reduced to 10 to 15 pct.

The "caviar" forming stage would generally require an additional 10 min. agitation in a test where 8 min. constituted the normal amalgam stage. Impellor speed in the apparatus is reduced from 1200 rpm to one-half or 600 rpm.

With the success of the formation of "caviar" it was decided to attempt to carry this idea even further by forming the amalgam into larger sized pellets. With this in mind, a churn or slow moving agitator was built. It consisted of a rectangular shaped steel box of 5-gal capacity when half full, and so placed that the horizontal drive shaft passed through the diagonal corners. This caused material in the box to run from end to end as well as turn in a circular motion. Rotating speed was 24 rpm.

In the application of the above apparatus, it was found desirable in order to speed up the formation, to first prepare the amalgam in the high speed agitation machine and then transfer the contents to the churn. An hour's churning turned the amalgam into small balls about the size of marbles. Longer periods of churning produced balls of greater diameter. Six hours of churning made balls 2 in. in diam. These larger balls contained but 5 pct moisture. Balls ½ in. in diam contained 8 pct moisture. By moisture is meant "surface" moisture.

These amalgam balls were hard enough to permit handling on screens and because of the squeezing out of clayey water were somewhat lower in ash than ordinary amalgam. They were not of sufficient strength, however, to permit handling in conveyors or storage into bins. When carbonized at low temperatures of 1000°F they could possibly make a satisfactory smokeless fuel.

Whether the use of bulk oil flotation has any commercial value is rather difficult to say. The cost of using 15 or 20 pct oil is normally considered prohibitive. Four or five Trent plants have been built in this country in past years but to the author's knowledge none has become a commercial success. Perhaps by producing a product similar to that made in the above churn and then making a low temperature coke, utilizing some of the by-product oils to make the amalgam, it may become an economic possibility.

Table 4 gives analytical data concerning Dorr Thickener overflow on which tests were made by bulk oil flotation to illustrate the effectiveness of this process for cleaning extreme fines. Dorr overflow was used because of its being 84.8 pct −200-mesh.

TABLE 4—*Champion No. 1, Dorr Overflow Solids*

Size	Pct Wt.	Pct Ash	Pct Sul.
48-mesh			
48 × 100	3.2	9.3	
100 × 200	12.0	10.6	2.00
−200	84.8	23.7	2.45
Head	100.0	22.4	2.40

Table 5 gives the analytical results of bulk oil flotation on the material shown in Table 4. An exceptionally clean concentrate is obtained of 5.2 pct ash at a 76.8 pct yield. Even sulphur reduction is remarkable. This reduction in sulphur is explained by the fact that overflow solids are recirculated solids which have passed many times through the washery water system and have become thoroughly oxidized. This test also illustrates the high amount of recoverable low ash coal available.

Reagent used in producing the amalgam consisted of 20 pct Pennsylvania crude petroleum oil and lime at the rate of 2 lb per ton feed.

TABLE 5—*Bulk Oil Flotation on Champion No. 1 Overflow*

Product	Pct Ash	Pct Sul.	Pct Wt.
Head	22.4	2.40	100.0
Amalgam	5.2	1.65	76.8
Tailings	79.8	4.75	23.2

SUMMARY

This paper describes some of the results obtained in coal flotation research at Champion No. 1 of the Pittsburgh Coal Co. It has been shown that bituminous fine coal can be readily cleaned by flotation methods and that it is possible to duplicate in plant practice most of the theories and facts demonstrated in the laboratory.

The author discusses certain aspects of flotation theory especially in regard to the new types of reagents now on the market and the importance of proper pH control. The bulk of the investigation work was conducted on Champion No. 1, Dorr thickener underflow slurry consisting mostly of particles under 48-mesh of which 33.3 pct was −200-mesh. This slurry ran 17.0 pct ash and 2.95 pct sulphur.

The cleanest concentrates obtained with this type feed was 3.6 pct ash at a yield of 52 pct. This required six re-cleanings of the concentrates. The economical range for Champion No. 1 slurry was 7-9 pct ash giving 80 to 88 pct yield. To achieve this would require re-cleaning the rougher or first concentrates at least once.

In coal froth flotation, the author found

that coarser particles float last, middle size particles down to approximately 20 microns will float first, followed by extreme fines of a colloidal nature, some of which will float with difficulty. Although it is possible to float particles up to 3⁄16 in., in practice it is much better to confine flotation to −28- or −48-mesh.

Most desirable pulp density for −48-mesh coal was found to be 10–12 pct solids. Higher densities result in higher ash concentrates, lower densities cause excessive reagent consumption.

Effect of moderate amount of soluble salts appears to be negligible in coal flotation. Some retardation appears, however, when using water containing a high amount of soluble salts. Alkalinity or acidity of the water in terms of its pH value affect the floatability of both coal and pyrite.

On the acid side, a better grade of coal as to ash content is obtained, but there also results a higher iron sulphide content. The highest coal recovery was obtained in a pH range of 7.0 to 7.5. With increase of pH value, recovery and grade dropped appreciably. Since higher pH depresses iron sulphides when lime is used as a depressor, in practice a pH range of 8.0 to 8.5 was held advisable.

The use of frothers, collectors and depressing agents are discussed. For Champion No. 1 practice it was found that the most suitable agents and their approximate consumption are as follows:

2 lb lime (CaO) per ton feed (dry basis)
0.45 to 0.65 lb Cresylic Acid
0.45 to 0.65 lb kerosene

It is carefully pointed out, however, that each operator must study his own particular needs and that many factors will affect not only the type of reagents used but the quantity consumed.

Attention is called to the desirability of using a frothing agent which will produce a brittle, rather than tough froth, in order to permit it to break down readily after leaving the flotation machine. It was found that kerosene used as a collector aided in this action. Fuel oil and Pennsylvania crude petroleum were also effective.

Although many depressors of iron sulphides were found effective in the laboratory, in plant practice no depressor appeared to be entirely satisfactory due to the tendency of pyrite or marcasite to become entrapped in the large volume of froth. Other agents were more effective than lime but the latter was used chiefly because of its cheapness and use in controlling pH value. Recleaning the concentrates was found necessary regardless of what depressors were used.

Iron sulphide can be either of the marcasite or pyritic variety. Evidence appears to support the belief that the marcasite variety is more difficult to depress than the pyritic. In either case, however, it seems that the oxidation products of oxidized or partially oxidized iron sulphides have a far more depressing effect than any outside agent used.

The investigation of bulk oil flotation is described. This particular type of flotation is especially effective in removing coal from clay and tests indicated that it did a better job on extreme fines, such as −200-mesh, than could be obtained by froth flotation. By this method, Dorr overflow water with solids running 84.8 pct −200-mesh and 22.4 pct ash was cleaned to 5.2 pct ash at a 76.8 pct yield.

For bulk oil flotation, crude petroleum, kerosene, fuel oil, water gas tar, and many crude or rough distillate cuts were effective in producing the desired amalgam, requiring, for good results, 15 to 30 pct reagent.

Slower churning of the amalgam after once formed produced pellets of coal and oil, which made them easier to handle and dewater.

There are numerous detrimental factors in the use of bulk oil flotation, however. This method of operation picks up all carbonaceous material, and where there is a high percentage of bone coal or middlings, these particles will be gathered with the recovered coal. This is also true of sulphides present unless they are thoroughly oxidized. Other serious objections are the high cost of reagents and a satisfactory method of handling the concentrates (amalgam). It may be possible to devise a method of carbonization whereby the reagent can be distilled off and re-used and a low temperature coke formed from the residue.

In presenting this paper, the author hopes that it may create a further interest on the part of others to conduct more research on the various factors affecting the flotation of coal. He believes that although the time is already at hand where some coal slurries and extreme fines may be cleaned economically by flotation, much work is yet to be done in applying the present day scientific tools available to coal technologists.

ACKNOWLEDGMENT

The writer is indebted to Mr. J. B. Morrow, under whose direction this work was carried out, for permission to present the information contained in this paper. Of various individuals on the staff of the Pittsburgh Coal Co. who have aided in the research work described in this paper, the author wishes to mention particularly Messrs. M. J. Sayers, J. T. Crawford, D. H. Davis and J. A. Younkins.

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DISCUSSION

(*T. W. Guy and J. Griffen presiding*)

M. G. DRIESSEN*—I would like to congratulate Mr. Zimmerman on this excellent paper and I would like to state that I have just come over from Holland where there are the largest coal flotation plants in the world.

The capacity of one mine is 10,000 tons a day and approximately 20 pct of the coal, 2000 tons a day, will be recovered by froth flotation. All this material is under ¾ mm.

Froth flotation is accomplished by mechanical stirring, and the process operates in about five successive stages. We do not use oil but we use the waste water from the cokeries.

I may add that the bituminous coal which we use is very easy to froth flotate and is a very excellent coking coal. I might ask Mr. Zimmerman when he speaks about inherent ash, what is the exact definition of inherent ash? As soon as the coal is broken up into sufficiently small particles, for instance under 50 microns, it can be cleaned down to even a few tenths of a per cent ash and so I think the expression "inherent ash" is most indefinite. It depends on just how far you go down by breaking it into smaller particles and separating it into different specific gravity fractions.

The low ash content, by separating the coal at about 1.28 cannot be obtained by ordinary ways but must be effected by centrifugal means.

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* Pittsburgh, Pa.

J. GRIFFEN—With the coals in northern France and Belgium—and I imagine those in Holland are the same—you may float the sink test and get a float coal of $^8/_{10}$ pct ash. That is very unusual in this country under the same conditions. Your general principle may be right but I doubt that our coals go down to quite as low ash, as far as the Pittsburgh scene is concerned anyway.

Mr. Zimmerman, I think it should be emphasized that this work was done quite a few years ago when coal was not worth as much as it is today. The economic angle had a great bearing on the particular lines which were followed in the laboratory work.

With today's value of coal, I think Mr. Zimmerman would agree that the results of his investigation could be reviewed and the emphasis might be on different techniques or reagents. Furthermore, the optimum procedures and reagents developed by Mr. Zimmerman for their Pittsburgh seam coal were largely influenced by the presence of considerable sulphur in the pyritic form, whether it is pyrite or marcasite. With present day economic considerations and with the absence of the sulphur problem, it is possible that different reagents might be selected for optimum economic results.

R. E. ZIMMERMAN (author's reply)—In answer to Mr. Driessen's interesting comments, it is well known that flotation of fine coal is practiced considerably in Europe. Noting his statements that one mine produces 2000 tons a day of minus ¾ mm, this would indicate that they must have a considerable fines problem as ¾ mm is approximately 20 mesh.

I believe my paper emphasized that the reagents used at Champion may not be necessarily suitable elsewhere and each individual plant must decide for itself what is best suited for its own local use, depending upon coal characteristics, water conditions and cheapness of available reagents.

As to an exact definition of "inherent" ash," I agree with Mr. Driessen that the line of demarcation between "inherent" and "extraneous" ash, as in the same line in determining coal moisture content, is quite difficult to obtain and is usually an arbitrary term. I ordinarily think of inherent ash as the lowest ash obtainable by float and sink methods on the sizes of coal normally treated by commercial cleaning plants.

In regard to Mr. Griffen's remarks that present coal values or economic conditions would warrant different techniques and different reagents, I do not quite follow.

In the first place, I specifically stated that there was no such thing as permanently picking a flotation reagent, but that newer and more reagents were continuously coming out on the market which could readily supplant those tested.

We were greatly concerned with sulphur reduction at the time of our research. I am sure a discussion with the steel people today would bring out the fact that it is an ever increasing problem now with the depletion of low sulphur coals.

The fact that coal is worth more today than it was before the war, in my opinion is an even better reason for following the lines suggested in my paper. A great deal of recoverable coal is being thrown away in refuse piles and nearby streams by most of our washeries and this could be recovered at a profit if treated by flotation.

I firmly believe that many of the basic fundamentals found in our research and reported in my paper are just as important today as they were before World War II and that the techniques used will be of considerable value to those continuing research in flotation. This statement is not to be misconstrued into believing that no further testing and experiments are necessary. Far from such a conclusion, it is readily acknowledged that our research was but a beginning and that much work has yet to be done.

Cleaning Anthracite Silt for Boiler Fuel with Humphreys Spiral Separator

By W. L. Dennen* and V. H. Wilson,* Members AIME

(New York Meeting, February 1948)

This paper is a description of the operation and results of a Humphreys Spiral Silt Cleaning Plant at the Powderly Colliery of The Hudson Coal Co. during the first nine months of operation and follows a preliminary description given at the Anthracite Institute in May, 1947, in a paper titled "Preparation of Anthracite Silt for Boiler Fuel in a Humphreys Spiral Test Plant" by H. H. Otto, V. H. Wilson and W. L. Dennen.

This plant is considered a full size operating test plant and, like all new plants, numerous minor changes in the methods of operation have been made from time to time to secure the results desired. Views of the silt preparation plant and of the screening house are given in Fig 1 and 2.

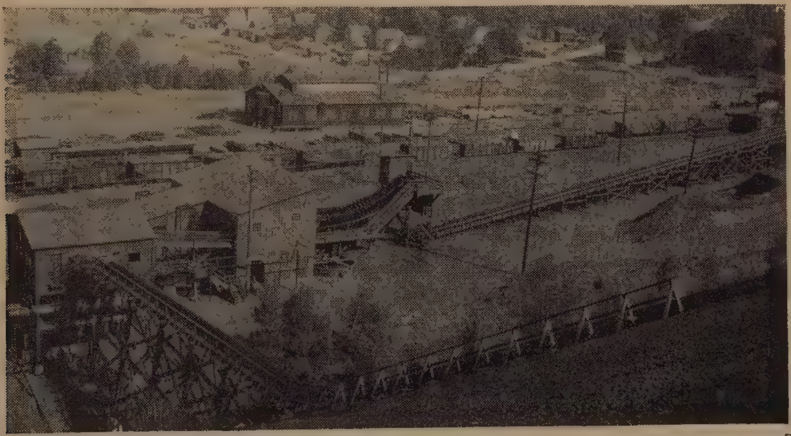

Fig 1—General view of silt preparation plant. Left is screening house. Toward the right the special separator plant, clean coal dewatering conveyors and belt conveyor in order.

This plant consists of 48 spirals in 8 batteries of 6 spirals each, 36 of which were to be operated as primary spirals and the remaining 12 as secondary spirals. A brief description of the plant and its operation follows. The only material cleaned at the plant during 1947 was that carried in the underflow from the Thickener which handles the Breaker waste water. This underflow, amounting to approximately 2000 gpm of water and containing 12.6 pct of solids, is pumped to vibrating screens where the plus $\frac{3}{16}$-in. material is removed. (Fig 3). This oversize material drops into a pocket from which it is trucked to the run-of-mine tipple for re-preparation.

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* Hudson Coal Co., Scranton, Pa.

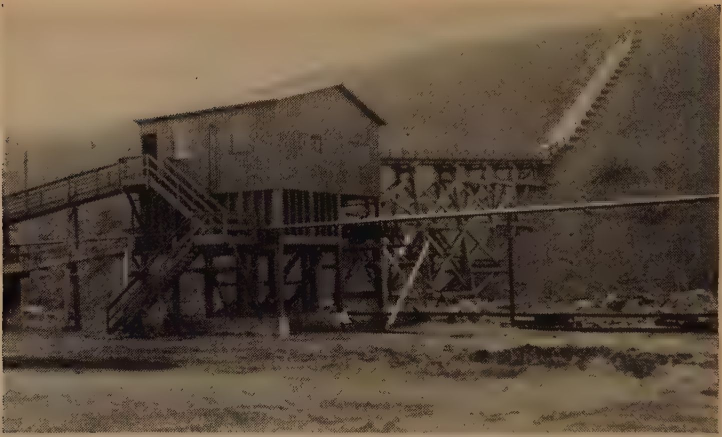

FIG 2—THE SCREENING HOUSE AND SILT DAM FROM WHICH THE SPIRAL PLANT WILL DERIVE MOST OF ITS MATERIAL.

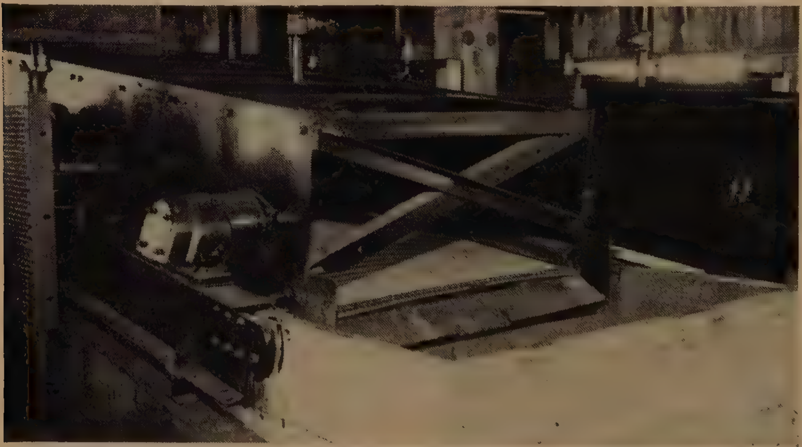

FIG 3—TWO VIBRATING SCREENS REMOVE THE OVERSIZE FROM THE FEED TO THE SPIRAL PLANT.

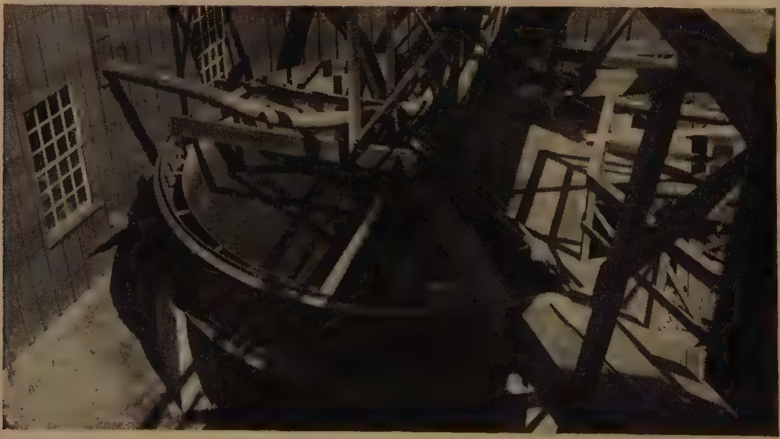

FIG 4—THE FEED TANK IN WHICH THE SLURRY IS AGITATED BEFORE BEING PUMPED TO THE SPIRALS.

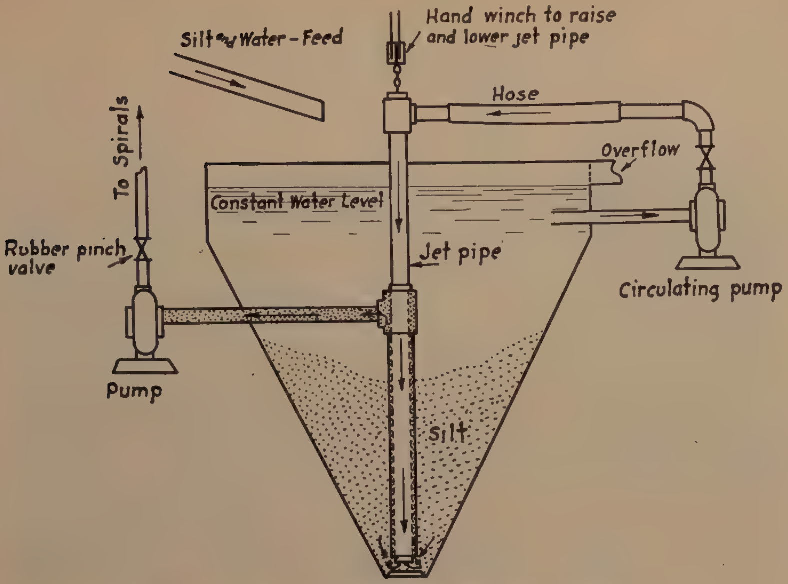

Fig 5a—Diagram of feed tank for spiral separators.

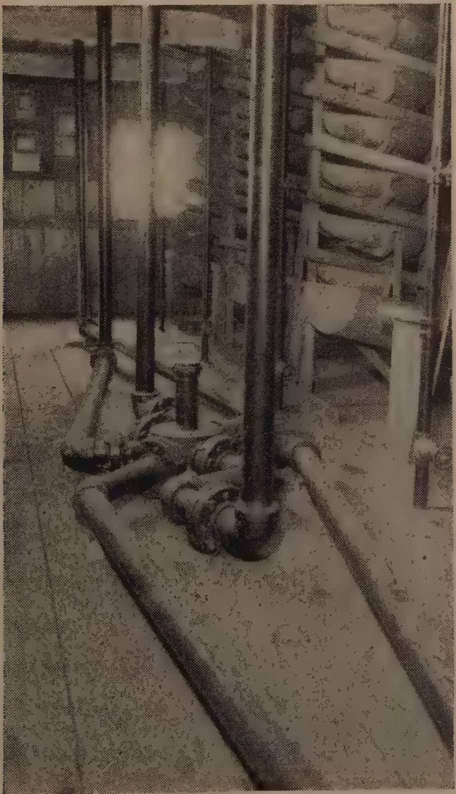

Fig 5b—The distributor on the floor equalizes the feed to the spiral batteries by means of elbows in the lines.

The fine material which is to be cleaned in the spiral plant drops through the screens and is flumed to a conical-shaped feed tank. An overflow of approximately 900 gpm is maintained at this tank to eliminate a large portion of the −200 mesh containing approximately 46.0 pct ash. This overflow is flumed directly to the refuse tank. The feed tank, Fig 4, is equipped with a piping arrangement to regulate the density of the slurry or pulp. This consists of two vertical pipes, one within the other, extending downward to within a few inches of the bottom of the tank, the inner or jet pipe being adjustable to height.

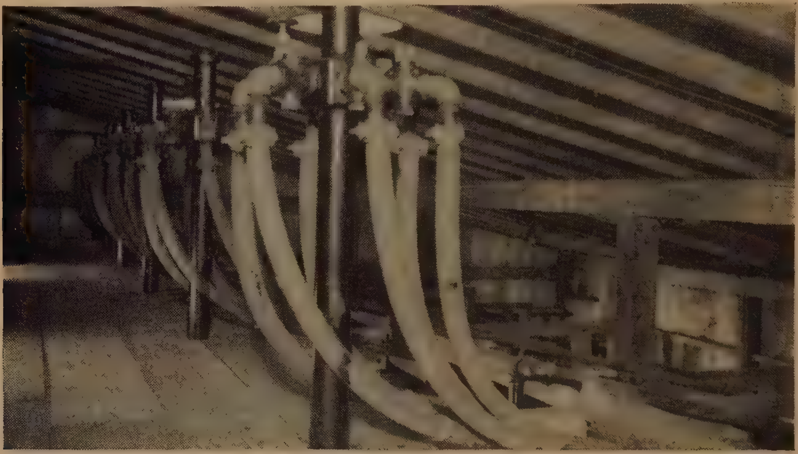

FIG 6—THE DIVIDER AT THE TOP OF EACH BATTERY OF SPIRALS DISTRIBUTES THE FLOW TO EACH SPIRAL IN THE ROW.

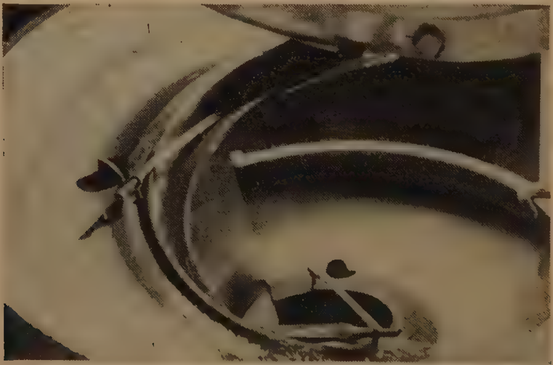

FIG 7—COMPLETE SPIRAL TURN SHOWING SPLITTERS AND FRESH WATER SCOOPS.

Water is pumped through the jet pipe to agitate the slurry, while the space between this pipe and the outer one serves as a suction for the feed pump which delivers the slurry to the centrally located distributing equipment, where the feed is split evenly to each battery of spirals (Fig 5a and 5b). 16–18 pct solids in the feed to the spirals proves to be the best mixture.

At the head of each row of the spirals the distribution lines end in dividers (Fig 6), with six-way outlets for splitting the flow so that a uniform quantity of slurry is delivered to each spiral.

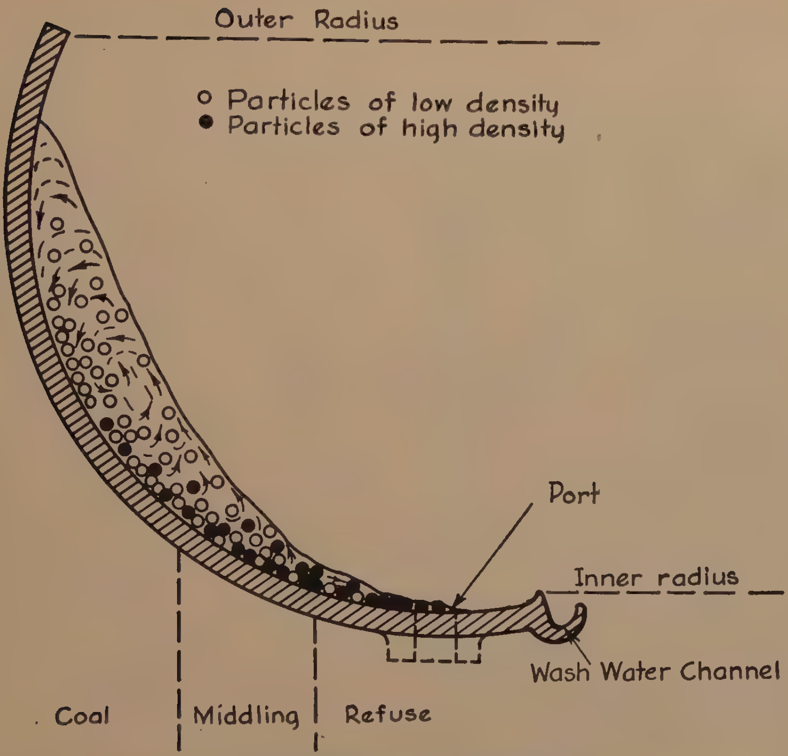

FIG 8—MODIFIED SECTION OF SPIRAL.

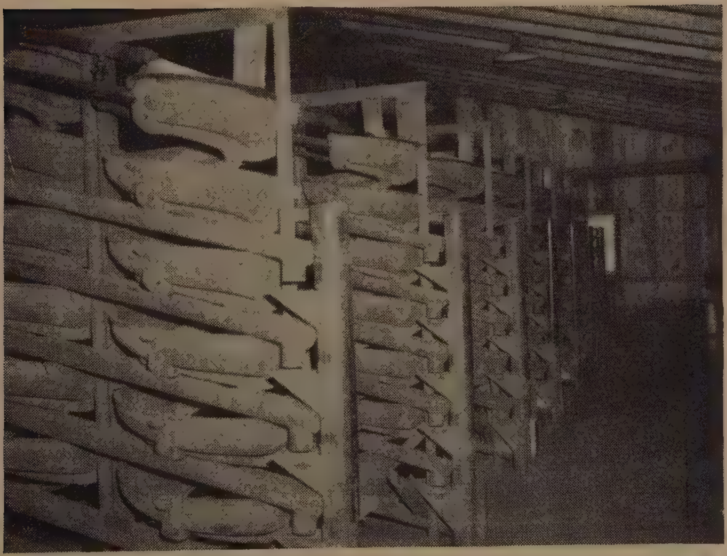

FIG 9—SILT IS CLEANED FOR BOILER FUEL IN THESE SPIRAL SEPARATORS MOUNTED IN BATTERIES OF SIX EACH.

The spiral consists of a cast-iron conduit or trough with a modified semicircular section, as shown in Fig 7.

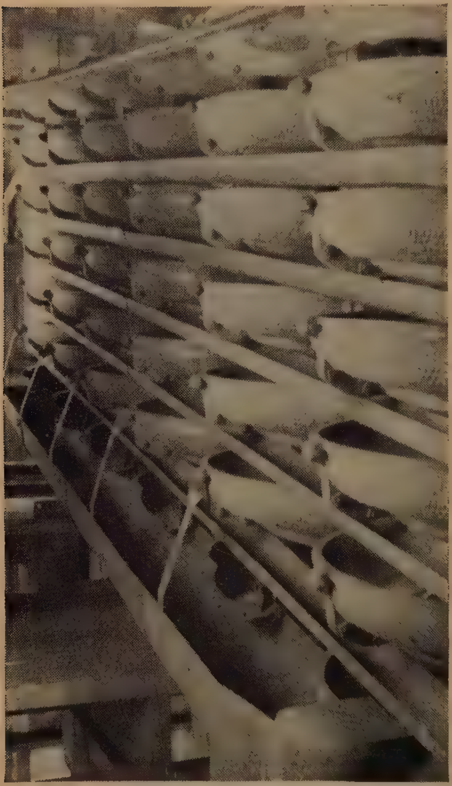

FIG 10—LOOKING FROM LOWER TOWARD UPPER END OF A BATTERY OF SPIRALS; COAL LAUNDER IS AT BOTTOM.

The sections are bolted together, three sections being required to complete a full turn. In the Powderly plant the spiral consists of six full turns on a 10-in. pitch with a diam of 24 in. Along the inner rim of the spiral is a small trough, as shown in Fig 7, down which clean water flows. The water is deflected by means of small scoops to the inner edge of the slurry stream, to assist in moving the high-gravity pulp into the flow. The high gravity material in the pulp follows the inner side of the spiral while the low gravity material travels along the outer edge, as shown in Fig 8.

In the case of anthracite, the high gravity material is the refuse and the light gravity material is the clean product. Heavy material, middling or refuse, drops through circular ports to refuse or middlings launders, while the coal continues down the spiral and drops to a launder leading to the dewatering tanks (Fig 9 and 10). One refuse port is located at each turn of the spiral and the setting of a splitter, seated in a counter-sunk recess, controls the character of material dropping through the ports.

The ash content in the clean coal and refuse can be increased or decreased, as may be desired, by adjusting the splitters. Once the splitters have been set to give the desired results they need not be changed again unless the percentage of ash in the feed changes very materially. A wide cut of the pulp at the ports results in lowering the percentage of ash in the clean coal and increasing the percentage of coal in the refuse, while narrowing the cut increases the ash content of clean coal and the refuse.

During the short time the plant was operated in the fall of 1946, we proceeded, according to plan, with 36 spirals as primary cleaners. The middling from the lowest port of each spiral was returned to the primary tank for recleaning and all refuse from the other ports was flumed to the secondary tank, from which it was pumped to the 12 secondary spirals for rerunning to make an additional coal recovery.

However, we found that very little additional low-ash coal was being recovered from the secondary spirals or from middling, and that the final refuse contained too low a percentage of ash. Therefore, when operations were resumed in April of 1947, we rearranged the piping and chutes for a one stage operation and are now pumping the material directly from the feed tank to the 48 spirals, as shown on the flowsheet (Fig 11). All of the material which drops through the spiral refuse ports is flumed to a conical-shaped refuse tank.

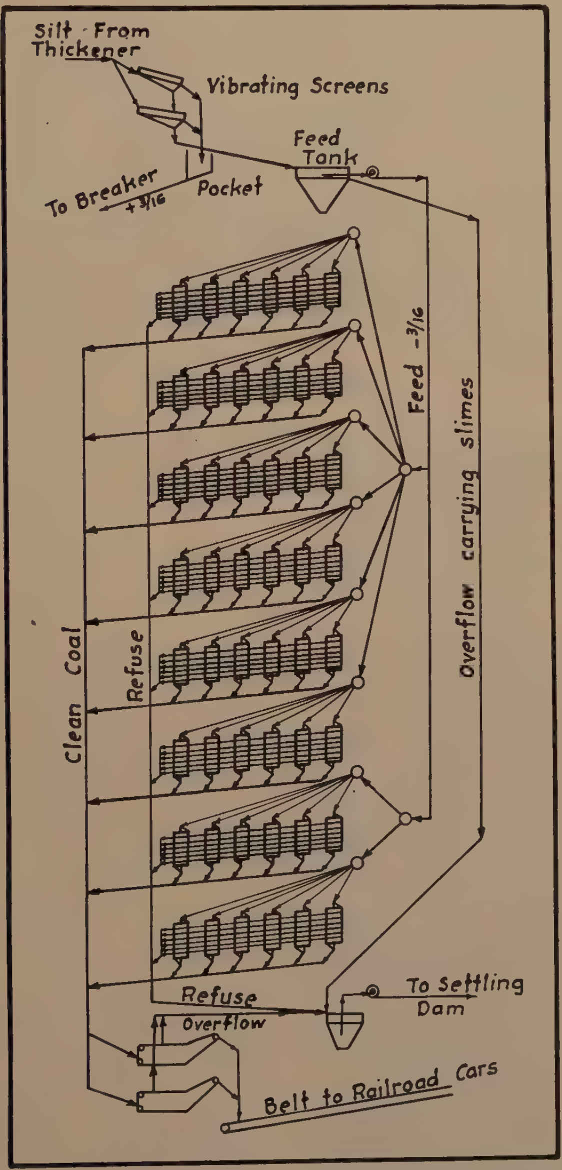

Fig 11—Flowsheet. Powderly silt cleaning plant.

From April 17 to September 30 of 1947 the ash content of the spiral-cleaned coal varied from 10.8 to 21 pct, and the ash in the refuse ranged between 64.5 and 86 pct. During the same period the ash content of the feed varied from 19 to 41.5 pct. Fig 12 shows the variations in the ash content of feed, clean coal and refuse for the period April 17 to September 30. An efficiency of

65 pct coal recovery was obtained in the operation of the spirals.

The clean coal from the spiral plant is flumed to settling tanks of wood construction equipped with slow-moving, chain-type dewatering conveyors with steel flights, which carry the coal to belt lines discharging into railroad cars (Fig 13–14). The overflow water carrying a small percentage of slimes is flumed to the refuse tank.

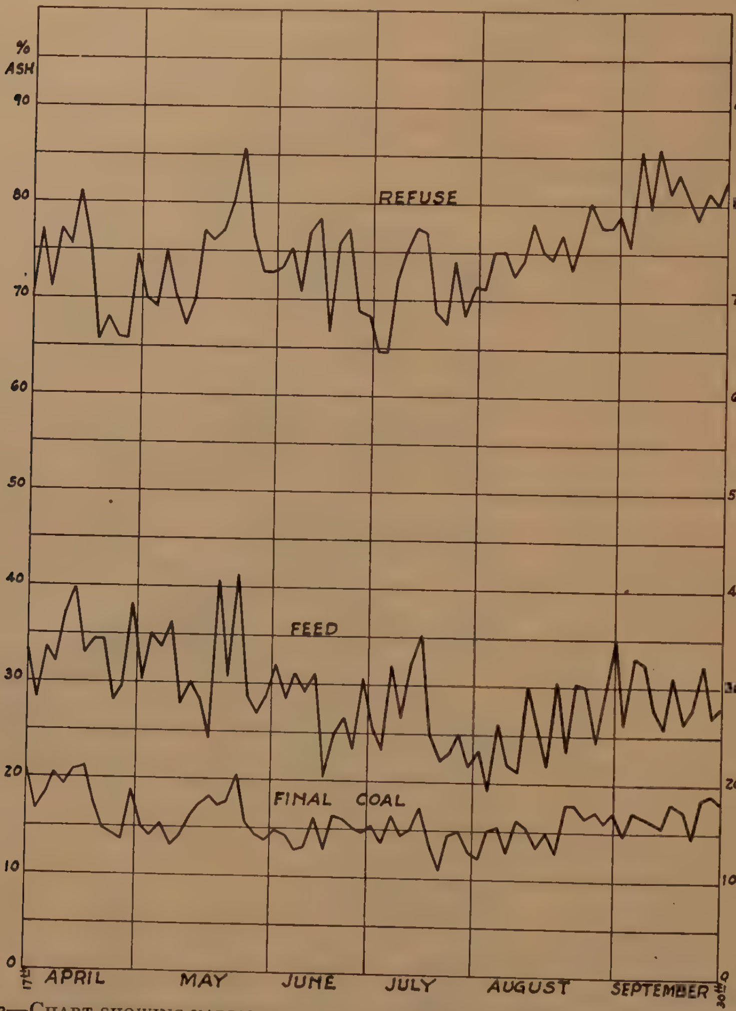

Fig 12—Chart showing variation in ash content of feed, refuse, and clean coal.

The contents of the refuse tank, Fig 15, are pumped to a refuse settling dam, Fig 16. Here the solids settle out and the water now seeps underground to a recently abandoned mine. However an outlet has been provided from the dam to a 16-in. borehole and, when overflow occurs, a suitable overflow control will be provided and the water permitted to drop down the borehole to the mine. When the mine becomes filled, the water will flow by gravity

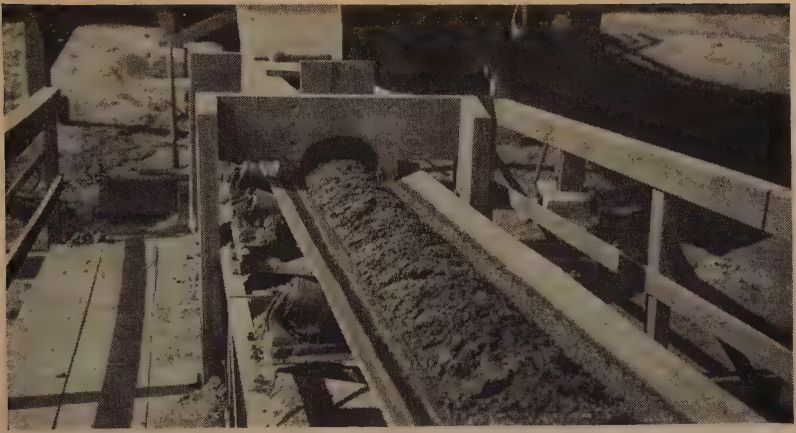

FIG 13—CLEAN SILT BEING DISCHARGED ONTO PRIMARY BELT LINE FROM THE DEWATERING CONVEYOR.

to daylight through an opening 7 miles distant. No dirty water from the Spiral Plant gets into the streams, and the system of settling complies fully with the pure stream laws of the State of Pennsylvania.

The only moving parts operating in the Humphreys Spiral Plant proper are three 6-in. pumps. One, having a capacity of 1000 gpm agitates the slurry in the feed tank, one of 1300 gpm lifts the slurry from the feed tank to the dividers at the head of the rows of spirals and a third, of 2300 gpm capacity, pumps the refuse from the refuse tank to the refuse settling dam. However, in addition, vibrating screens, clean coal dewatering conveyors and belt lines are necessary for the complete operation, as mentioned above.

Fig 17 shows typical quantities of solids and water at various points of the system.

The boiler furnaces at the Coal Brook Power Plant were designed to burn anthracite silt on traveling grates and have operated satisfactorily with fuel having an ash content ranging from 13 to 22 pct. During a period when the boilers had to be worked at their maximum capacity due to an unusually heavy mine pumping load we burned fuel from the Spiral plant running all the way from 12.5 to 17.0 pct ash. The

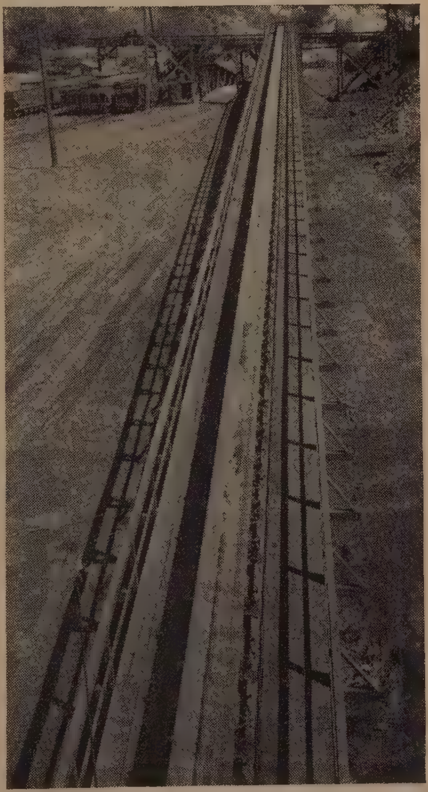

FIG 14—PRIMARY BELT LINE WITH SECONDARY BELT AND RAILROAD CAR LOADING STATION IN BACKGROUND.

silt now being produced at the Humphreys Spiral Plant is averaging approximately 18 pct ash and has proved to be a very satisfactory fuel. Apparently the surface of the product is comparatively free from film or scum so that the individual particles of coal ignite rather freely on the grates.

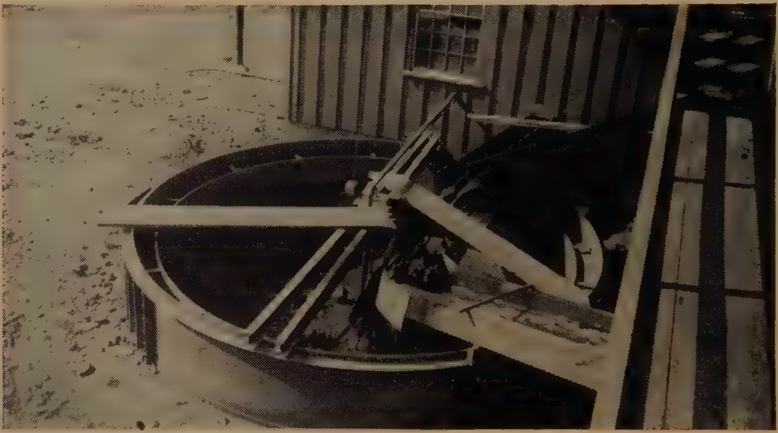

FIG 15—REFUSE TANK SHOWING LAUNDERS CARRYING OVERFLOW FROM PRIMARY TANK, COAL DEWATERING TANKS AND REFUSE FROM SPIRALS.

FIG 16—A PARTIAL VIEW OF THE REFUSE SETTLING DAM. THE INLET ON THE RIGHT AND THE OVERFLOW POINT ON LEFT ARE BOTH OUTSIDE THE PHOTOGRAPH.

Tables 1, 2 and 3 show size and ash analyses of typical spiral feed and prepared fuel.

Powderly Breaker, during the period between April 14 and September 30, 1947, operated 135 shifts and produced 464,795 tons of prepared coal, including that cleaned in the Humphreys plant. During this period 30,314 tons of boiler fuel were produced from the material which was carried to the Dorr Thickener in the breaker waste water. All of which would have been pumped to the silt dam if there had been no silt cleaning plant.

The spiral plant is operated by one attendant and the only other help employed in connection with the plant is a colliery railroad car loader, who spends part of his time moving the silt cars. The maintenance of the plant is very low and the power costs per ton have been slightly over 2 ¢ per ton of fuel produced. If and when the plant is operated to maximum capacity it may be necessary or desirable to add another man to operate it. However, the maintenance and power costs per ton will change but little.

Thus far the spiral plant has been operated only on the day shift and is now (October 1947) producing an average of 320 tons of clean silt per start. A drag scraper gathering system is being installed to col-

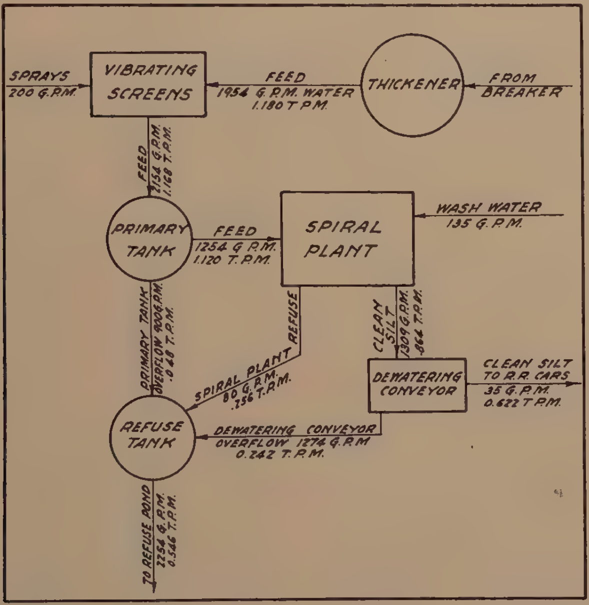

FIG 17—SKETCH SHOWING TYPICAL FLOW IN VARIOUS PARTS OF THE HUMPHREYS SPIRAL PLANT IN GALLONS PER MINUTE OF WATER AND TONS PER MINUTE OF SOLIDS.

TABLES 1, 2 AND 3—*Size and Ash Analyses of Samples Taken on Three Different Operating Days*

Mesh	Table 1				Table 2				Table 3			
	Feed		Clean Coal		Feed		Clean Coal		Feed		Clean Coal	
	Per Cent Weight	Per Cent Ash	Per Cent Weight	Per Cent Ash	Per Cent Weight	Per Cent Ash	Per Cent Weight	Per Cent Ash	Per Cent Weight	Per Cent Ash	Per Cent Weight	Per Cent Ash
+3/32 in.	6.6	26.0	11.6	14.0	5.0	25.0	8.4	17.5	8.8	20.3	10.2	19.7
+1/16 in.	12.4	21.3	20.2	10.5	12.8	20.4	19.6	12.4	8.4	20.6	9.2	15.0
+3/64 in.	16.6	20.5	24.2	10.2	12.8	20.7	18.6	13.6	17.2	19.7	21.2	14.8
+ 28	11.8	22.0	15.0	9.7	14.4	23.2	18.8	13.7	14.8	21.4	16.6	13.0
+ 35	9.8	23.8	10.0	10.1	10.6	23.3	11.4	11.7	14.2	24.2	15.0	14.0
+ 48	9.6	32.5	7.6	11.4	10.4	40.8	9.2	12.0	11.2	33.7	11.2	16.0
+ 60	8.0	42.5	4.2	17.6	9.0	50.3	5.6	10.8	8.2	49.8	6.4	20.3
+ 80	5.4	49.0	2.0	21.0	6.8	50.5	3.0	20.4	6.2	64.2	3.4	31.4
+100	4.2	49.9	2.0	36.9	4.2	49.0	2.0	37.1	3.2	60.3	2.4	42.0
+200	8.4	49.6	2.2	60.6	7.6	64.5	2.4	53.0	4.6	59.8	2.8	60.0
−200	7.2	60.5	1.0	68.2	6.4	57.5	1.0	62.7	3.2	63.1	1.6	70.2
Comp. Ash..........		32.4		13.3		35.2		15.3		32.0		19.4

lect the material in the silt dam and it is planned to operate the plant two 7-hr shifts each day for the eight months of nonfreezing weather during the year. It is estimated that we will produce 650 tons of clean silt each 24 hr day, with the plant operating on breaker material plus some silt dam material during the day shift and on silt dam material alone during the other shift. A full year's supply of fuel for the Coal Brook power plant will be provided by this arrangement.

Cleaning anthracite silt by means of Humphreys Spirals has proven successful for our purposes. However the plant is still considered an experimental operation and during the winter shut down, some minor improvements are to be made. They will include the replacing of the refuse chutes with vitrified clay launders, and replacing the plain steel feed line from the Thickener sludge pump with rubber-lined steel pipe, which was not available when operations were started.

The authors wish to express their appreciation to Messrs. H. H. Otto, Asst. Gen. Mgr., in Charge of Engineering, John S. Johnson, Supt. of Preparation and Maintenance, W. E. Thompson, Engineer, and others of the engineering and laboratory staffs of The Hudson Coal Co. for their suggestions and assistance in the preparation of this paper.

DISCUSSION

(*J. Griffen and T. W. Guy presiding*)

H. H. Otto*—As Mr. Dennen told you, we were confronted with the problem of getting some boiler fuel for one of our power plants. The Hudson Coal Co. has two major power generating stations which generate considerable power for our own use; the balance is bought as needed.

We were also confronted with a rather large commercial demand for fine sizes. Our coal supply in the Carbondale area is diminishing and we are beginning to get worried about the boiler fuel supply for our Coal Brook power plant.

At Powderly Colliery we have a big silt bank which probably contains a million and a half or more tons of material to be cleaned; and, as Mr. Dennen mentioned, we were cleaning in this spiral plant breaker silt with about 17 or 18 pct ash in the finished product. This percentage of ash may seem high to you but it is not bad for us. Our boilers were redesigned in order that we could burn high-ash fuel.

Formerly we used a much lower ash fuel. One of my friends in the anthracite region told me that the best results could be obtained if we burned No. 3 or No. 4 Buck containing a 12 pct ash. I advised him that this material, in our case, was going to the market and we were now prepared to burn material which had been considered waste in years gone by.

The President of our Company, who was Vice President and General Manager in 1945, accompanied by our Preparation Superintendent Mr. J. S. Johnson, who is present here today, saw a spiral plant in operation, in South Jacksonville, Florida. I also saw the plant at a later date, and we came to the conclusion that the Humphreys Spiral had possibilities in cleaning the very fine anthracite silt.

No one had any experience with cleaning anthracite on the Humphreys Spiral and therefore we started, as it were, from scratch. The plant is not doing everything that we have a right to expect from it but, like any new venture of this type, adjustments and changes must be made and we have a lot to learn in order to get the best results possible.

The plant was started April 15th of last year and closed down about December 1st because there is no heat in the plant and we can fill our fuel requirements without winter operation. During the month of October, 1947, we prepared an average of 330 tons of usable boiler fuel in the spiral plant for use in our Coal Brook power plant. This was accomplished on a 7-hr shift per day. We took the finished product from the spiral plant and, of course, had other fuel from other sources to help build up the stock pile at Coal Brook to last us all winter. Stream pollution laws are being enforced by the State of Pennsylvania and, judging from reports from Washington, the rest of the country is going to be up against Federal stream pollution laws in the near future, which will have a bearing on the installation and use of fine coal cleaning equipment. This is based upon the report of a congressman who told me that there will be some legislation enacted and we should get ready to clean up our streams. Therefore efforts are being made to get all of the usable material out of the anthracite silt before the final waste is either sent into the mines or put in refuse piles.

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* Hudson Coal Co., Scranton, Pa.

This spiral plant assists us in taking care of the greater part of our stream pollution problems at Powderly Colliery. Our results at the Powderly Humphreys Spiral Plant have caused us to give favorable consideration to a plant at another colliery.

In the anthracite region today the Scranton Electric Co., Luzerne Co. Gas and Electric Co. and Pennsylvania Power and Light Co. are installing pulverizers for anthracite fuel boilers. This would indicate the desirability of cleaning the very fine anthracite silt because of the increasing market.

I think Mr. Dennen and Mr. Wilson did a good job in the preparation of their paper and I want to thank them personally for the great interest that they have taken in it. I also want to thank Mr. John S. Johnson for helping to make the plant a success.

As far as we are concerned it is a very economical plant. We had very few repairs the first year. Naturally after it has been in operation for a few years we will know more about maintenance costs.

In 1948 we are going to continue our experiment to see how low an ash we can make consistently. This will be done as needed. First, however, we will take care of our boiler supply needs where we use fuel with the higher ash content.

J. S. Johnson*—I do not know that there is anything I can add to what has already been said about the utilization of these spirals for making boiler fuel. We know from our experiments that we can also make a very low ash coal for the market.

When Mr. Evans and I went down to Jacksonville, Florida, to look at these spirals working on various minerals we found that they were working very successfully at 176 spirals located out there in an open space with only one man to supervise.

They were handling 6000 tons of sea sand and these minerals in 24 hr. The recovery was about 5 pct but that recovery of 5 pct was of minerals that aggregated about twelve different kinds. In order to make use of those methods in the anthracite field—that is in the line of recovery—we had to do just the opposite of what they were doing there. What they were recovering went down where our refuse is now going. I believe that the spirals will do the work on this fine coal.

There is one thing that we found out through our experimentation, that is, that the material should be more uniform and should not be of a very large variation of sizes. In keeping the sizes for the present pitch of the spirals we have now anywhere between the 3⁄32-in. mesh size and down through the 200. It makes a very good clean job.

I believe that a great many other companies will pick up these spirals and utilize them for the cleaning of their fine coal.

J. Griffen—Of course we all appreciate that this particular job on which the spirals have been used is a little different from the usual cleaning job. I was hoping perhaps you would make some comparisons with the cleaning results obtained in some of your other plants. At the installations at Marvine, I understand you were turning out a 10 or 11 pct ash product for shipment to market.

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* Hudson Coal Co., Scranton, Pa.

W. H. Lesser*—I would like to ask Mr. Dennen what is the velocity of the water coming down the spiral? Give me the feet per second or per minute.

W. L. Dennen (authors' reply)—I do not know but possibly Mr. Wilson can answer that.

V. H. Wilson—The velocity is about 6 ft a second.

W. H. Lesser—On what principle do you consider you make the separation? Does it go to the center and not to the end?

M. Welker†—The first principle of separation is that of a heavy mineral settling out of a flowing stream as you would have in an ordinary launder.

W. H. Lesser—If you have a particle of coal with a gravity of 1.7 and slate with a gravity of 2.3, does this affect the principle on which the separation of slate from coal depends?

V. H. Wilson—That is a factor. First of all, your slate will settle out of the material more rapidly than the coal. Then you will get it next to the surface of the iron and there, retarded by friction, it slides down the cross-section of the spiral to the lowest portion of the cross-section.

W. H. Lesser—Did you say 6 ft a second?

V. H. Wilson—Yes, that is about the maximum velocity. The stream varies through its cross-section.

W. H. Lesser—Did you ever try to double that speed and see what centrifugal force it has?

V. H. Wilson—We have increased it and decreased it some, and so far we think we have about the optimum.

W. H. Lesser—You do not know what the segregation would be if you made that 12 ft a second?

V. H. Wilson—No, we have not gotten quite that far. We have gone 8 or 9 ft a second.

W. H. Lesser—And still the principle under those conditions depends upon the fact that the heavy material is held up in the center and the centrifugal force does not enter into it at that speed?

V. H. Wilson—The centrifugal force is a minor thing there. The principal separation is due to the fact that the heavy material is slowed up by friction on the surface of the bottom of the spiral. The coal goes rushing on over and as it goes more slowly, the centrifugal force on the heavy material becomes nil.

W. H. Lesser—In other words, the principle is practically the same as the old Fahrenwald Separators of about twenty years ago?

V. H. Wilson—That may be.

Member—I would like to ask Mr. Johnson if, in the experimentation work, there was any coal that was actually of the nature of a 6 or 8 pct ash that they tried to bring down to the lower ash ratio?

J. S. Johnson—The material with which we experimented would run anywhere from 18 to 52 pct ash which we applied over these spirals; and if you get a 52 pct ash material, you do not get quite as good recovery in ash as you do with around 18 to 25 pct. That is, your volume recovered is less. I believe that holds good in almost any kind of machine in which you try to clean anthracite coal.

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* Pierce Management, Scranton, Pa.
† Humphreys Spiral Co.

M. G. Driessen*—Is the clean coal thickened before it goes to the boiler? What is the moisture content?

W. L. Dennen—The clean coal as it leaves the dewatering tank has about 35 pct water in it. By the time it gets into the boiler plant after it has been standing in the railroad cars there is about 15 pct water in it.

M. G. Driessen—Just what are the proportions of water to solid used in the process?

W. L. Dennen—We would like to get about 16 to 18 pct of solid in the feed.

M. G. Driessen—About 6 tons of water to about 1 ton of solid?

W. L. Dennen—Yes.

J. Griffen—What is the horsepower per ton with regard to the pumping and so on?

W. L. Dennen—The power used at this plant is not metered.

J. Griffen—What tonnage have you been feeding per hour?

W. L. Dennen—We feed about 1 ton per spiral per hour and that gives us our best results.

J. Griffen—Forty-eight to 50 tons an hour?

W. L. Dennen—Yes.

S. A. Stone*—I wonder if you have any data on recovery efficiencies? After all you are making a separation at a given specific gravity, and I wonder if you have any data on efficiency of recovery of the light gravity coal?

W. L. Dennen—We recover between 60 and 65 pct of the coal.

J. Griffen—You mean of the feed?

W. L. Dennen—Yes.

S. A. Stone—What per cent of the float coal available in the feed do you recover?

V. H. Wilson—We recover as high as 80 pct. That is 80 pct efficiency, and there is 17 pct ash in the coal.

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* Pittsburgh, Pa.

* The Deister Concentrator Co., Fort Wayne, Ind.

Grinding of Anthracite for Pulverized Fuel

By C. H. Frick*

(Philadelphia Meeting, October 1946)

Before presenting the main topic, as indicated by the title, this paper will give some of the high-spot history of the anthracite industry.

Introduction

The earliest recorded use of anthracite was in the year 1762, when a company of Connecticut pioneers discovered anthracite near the present city of Wilkes-Barre, Pennsylvania. In the early days of the industry, the coal was used only in large pieces, the smaller pieces being discarded to waste piles. As the output of the industry increased during the nineteenth century, uses were developed for somewhat smaller sizes down to and including pea coal. Early in the twentieth century, sizes smaller than pea, such as No. 1 and No. 2 buckwheat, were developed and used. Accumulations of piles from current mining operations were still increasing; that is, the smaller sizes were still being discarded. It was not until after World War I that preparation of the so-called steam sizes began commercially. This was largely No. 3 buckwheat, which is screened through a 3/16-in. round-hole screen and over a 3/32-in. or a 1/16-in. round-hole screen.

Use of Smaller Sizes

The earliest recorded experience in burning the small sizes of anthracite for steam production, other than in a previous small experiment, was in the year 1913, when a size known as "boiler fuel" was introduced—a mixture of rice and barley (No. 2 and No. 3 buckwheat). At that time, there was still no method known for burning the still smaller sizes.

Table 1.—*Screen Sizes Used for Small Sizes of Anthracite*

Size of Coal	Industry Standard, In.	Purpose for Which Used by Boiler Plants
No. 2 buckwheat (rice)	−5/16 + 3/16	Hand firing
No. 3 buck. (barley)	−3/16 + 3/32	Hand firing Stoker firing
No. 4 buckwheat	−3/32 + 3/64	Stoker firing
No. 5 buckwheat	−3/64	Pulverized fuel
Pulverized		Pulverized fuel

At the present time, various sizes down to and including No. 5 buckwheat are being prepared (Table 1) from current mining and bank-reclaiming operations. The sizes of No. 4 and No. 5 buckwheat and smaller are generally being used for steam production in large boilers.

The Pennsylvania Power and Light Co., following World War I, used 680,000 tons of anthracite per year, burned largely on stokers for production of steam, whereas for the year 1943 it used 1,940,000 tons of stoker sizes and 400,000 tons of smaller sizes for grinding and burning in pulverized form. This company has taken a very large part in the research and development of ways and means of burning these smaller sizes of anthracite, including the grinding of the very small sizes. This paper is particularly concerned with the latter feature.

Listed for New York Meeting, February 1945, which was canceled. Manuscript received at the office of the Institute March 8, 1945. Revision received June 29, 1946. Issued as TP 2061 in Coal Technology, August 1946.

* Plant Betterment Engineer, Pennsylvania Power and Light Co., Allentown, Pennsylvania.

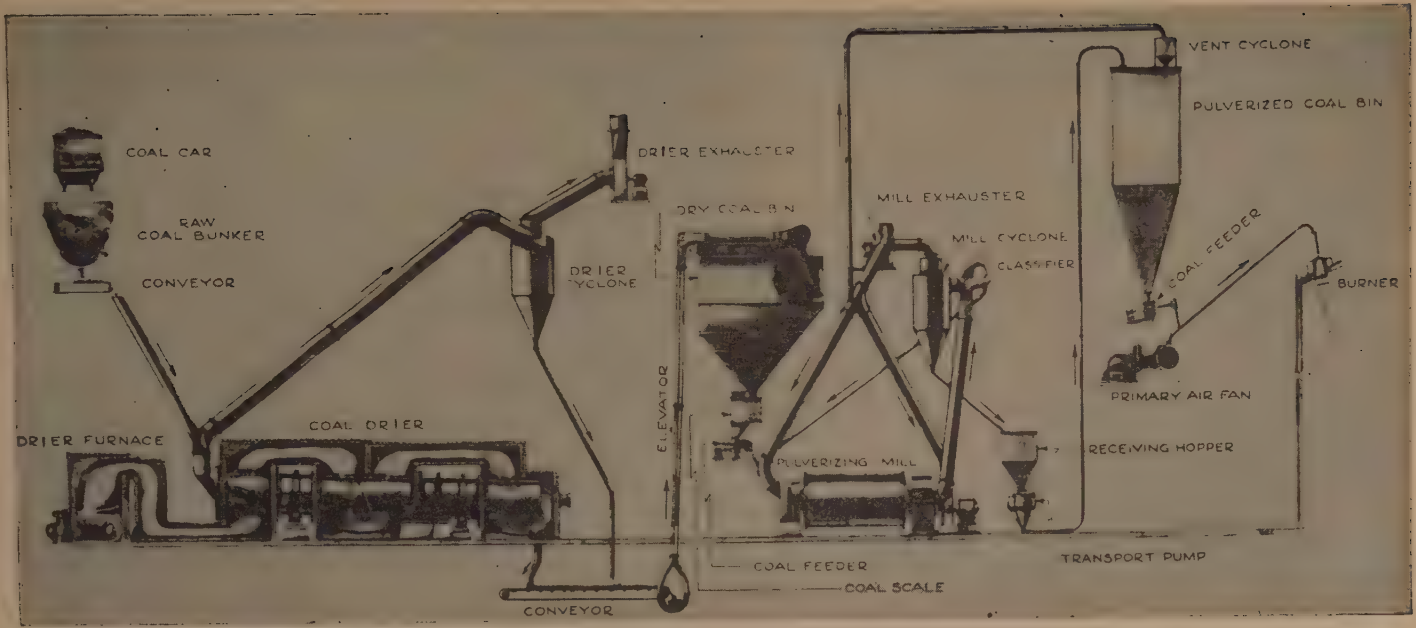

FIG 1.—ANTHRACITE PREPARATION PLANT FOR HAUTO STATION, PENNSYLVANIA POWER AND LIGHT COMPANY.

General Considerations

The object of grinding very small anthracite and burning in pulverized form, as compared with burning small sizes on stokers for steam production, is that a lower-cost fuel is available for pulverizing than can be used successfully on stokers.

Anthracite of all sizes is prepared—that is, de-ashed—by various processes, each one using water as one of its agents, and when the coal is loaded into trucks or cars for shipment, there is still a relatively large amount of moisture attached to the anthracite when it reaches its destination. In the winter this may reach 20 per cent or more.

Early Installations

The first known installation to burn anthracite in pulverized form was at the Lytle power plant of the Susquehanna Collieries Co. (an anthracite-mining company), in the year 1918. This was on a very small scale, however.

A second and larger installation was made at Lykens[1] by the same company and put into service in the year 1921. This consisted of a separate plant to dry and pulverize the coal, from which it was transported by mechanical conveyors to the boiler plant, which contained six 500-hp. boilers and six 600-hp. boilers. This plant was purchased by Pennsylvania Power and Light Co. in February 1942, and is being operated by that company. The anthracite used in these plants was smaller than No. 3 buckwheat, and was known as slush, about 70 per cent of which passes through a 3⁄64-in. round-hole screen. This material was the discharge from the near-by breaker, so small in size as to be unmarketable. It was dried and ground to a fineness of about 82 per cent through a 200-mesh screen. This coal contained about 8 per cent volatile matter and burned very satisfactorily in the furnace designed for it.

During the spring and summer of 1924, Pennsylvania Power and Light Co. arranged with Susquehanna Collieries Co. to conduct a series of tests in the Pennsylvania plant. It segregated one drier, one mill, one transport system and one boiler, and used coal from a number of different locations in the entire anthracite region.

These tests demonstrated that anthracite from any location could be satisfactorily ground and burned,[2] and as a result of this, the Pennsylvania Power and Light Co., which was then considering extension of its electric production facilities, decided to enlarge its Pine Grove plant, which is in the same general area as the Lykens plant, and burn pulverized anthracite there. The Lykens plant contained eight 42-in. screen-type vertical mills, whereas the Pine Grove plant[3,4] as designed contained three mills, each of 70-in. diameter, of the same type and make. After a few years' experience, these were replaced by ball and tube mills.

About the same time that the experiments at Lykens were concluded and the design of the Pine Grove extension was underway, another public utility company designed a steam-power plant to burn pulverized anthracite, using vertical ball mills.

Hauto Installation

Because of the business conditions of the 1930's, development work was delayed in connection with expansion of facilities to grind and burn anthracite. However, with the increased loads caused by the war, a contract was let for installation of pulverizing equipment and boilers at the Hauto steam electric station[4] of Pennsylvania Power and Light Co., with the result that the two largest boilers in the world to burn pulverized anthracite were installed and put into service in the summer of 1943 (Fig 1). This plant is similar to both the Pine Grove and the Lykens in general layout, although it is very much larger. The preparation plant consists of two cylindrical rotating driers, each 60 ft long by 7 ft 6 in. in diameter and having a rated capacity of 30 tons per hr at normal speed, from initial moisture of 20 per cent (10 per cent ice and 10 per cent water) to a final moisture of 1 per cent, and three horizontal ball and tube mills, each with a ball charge of 37½ tons when grinding coal with a moisture content of 1 per cent to a fineness of 85 per cent through a 200-mesh screen.

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[1] References are at the end of the paper.

DRYING EQUIPMENT

These driers normally operate at 2.5 rpm, but the two-speed motors that drive them can double the drier speed if desirable, and if moisture content of entering coal is sufficiently low to allow this increase in speed. Under this condition capacity may reach 60 tons per hour of dry coal.

The driers are heated by separate and specially designed stoker-fired furnaces, each equipped with two turbovane blowers for supplying forced draft. Stokers are of the traveling-grate type, each 7 ft 8⅝ in. wide and 11 ft 0 in. long. Furnace-gas temperature to the drier is automatically controlled by a specially designed tempering damper in the duct from furnace to drier setting, outside of the rotating drier shell. The hot gases travel through the drier setting outside of the drier shell and return through the inside of the shell, passing thence to adequate cyclones and exhausters to be vented to the main boiler stack; coal dust recovered from the cyclone separators is fed by gravity to the dry-coal-conveying system and is delivered with the coal from the driers to the bins ahead of each pulverizing mill.

Other types of driers are available for reducing moisture content of small anthracite: (1) the centrifugal, flash type, tower type, where coal is introduced at the top of cascades over a series of baffles, falling through hot gas arising from the bottom and (2) the steam-heated type. It is the writer's experience that the direct-fired rotating cylindrical type is very satisfactory for the purpose.

The amount of material carried out of the drier in the gas stream will be a function of the gas velocity at the drier exit, which is influenced by air leakage at seals, size of material being dried, and rotational speed of drier shell and method of feeding coal into drier. Gas velocity in turn varies with tons of material being dried, moisture extraction and diameter of drier shell.

MILLING EQUIPMENT

A typical horizontal ball and tube mill is illustrated in Fig 2.

In order to maintain maximum output of mills at desired fineness, which is most essential in a storage system, the coal feed to the mill should be as uniform as possible and the level of coal in the mill maintained at the best predetermined level.

The coal feed can be regulated by using an electrical device (an electric eye) working in conjunction with a draft connection at normal coal level in the mill.

A cyclone should be in the circuit ahead of the mill exhauster, to remove the finished product and reduce exhauster maintenance. A separator ahead of the cyclone, to remove the coarse particles, permits circulation of more air through the mill and thereby increases its capacity.

There is an optimum speed of a mill of a given diameter to maintain the maximum output and desired fineness. At the Pine Grove plant, several years ago, a number of tests were conducted with the speed of the mill varied, electrically segregating it with one of the house turbines so that the speed could be varied from 95 to 105 per cent of rated speed to determine the effect of speed on the mill capacity. No improvement in capacity resulted, but with low speed proper fineness was not obtained, while with high speed difficulty was experienced with overfeeding, causing the mill exhauster motor to trip out.

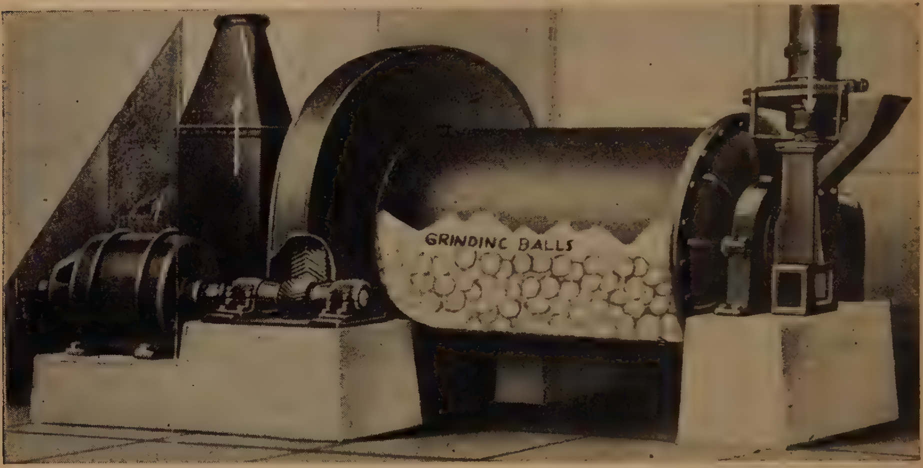

FIG 2.—TYPICAL HORIZONTAL BALL AND TUBE MILL.

Furthermore, in a preparation plant where the bin system is used, it is very necessary that the various functions be coordinated electrically, so that if the motor driving the apparatus near the outlet end of the coal circuit fails—for instance, transporting pulverized coal from the mills to the storage bins—preceding operations will not continue, in order to prevent blocking of the entire system. Accordingly, all of the controls are interlocked electrically, so that all operations will stop whenever failure occurs.

The type of cyclone used is also very important in the effectiveness of reclaiming material that may be lost through the various vents, and certain design factors must be carefully considered, such as ratio of height of cyclone inlet to height of inner cylinder and the relative position of each.

Mills for Grinding Anthracite

In the early years of grinding anthracite, the vertical ball mill was generally used. One small installation was made of a steam-jet mill, which consisted of a steel cylinder with a ring of nozzles at the top through which the coal passed vertically downward, and was pulverized by impact of the steam jets converging to center at very high velocity. This installation was made on a trial basis by a coal company and has since been discontinued.

Anthracite does not behave like bituminous coal or other materials that are ground with a vertical mill, in that the coefficient of friction between the various anthracite particles is very low, the particles having a tendency to slide on each other and slide off the ball race, whereas with the horizontal mill many tons of small balls are raised by the rotation of the mill and dropped, causing thousands of hammer blows, which crush the coal to the desired fineness. In the vertical mills, the few balls, usually 12 or 15 in. in diameter, tend to slide and lose their roundness, making necessary frequent replacement and high maintenance costs, whereas in the ball and tube mill the small balls, generally 1¼ to 1½ in. in diameter, wear down to almost nothing, and addition is made periodically of new balls to maintain a certain level in the mill in order to maintain capacity.

For the foregoing reasons, therefore, the horizontal ball and tube mill is the most used type of mill today for grinding anthracite. Its greater availability and lower maintenance cost per ton of coal ground make it more desirable than the vertical mill.

As mentioned earlier in this paper, experiments were conducted during 1924 at the Lykens plant with coal from various sections of the anthracite territory and it was found that there is considerable

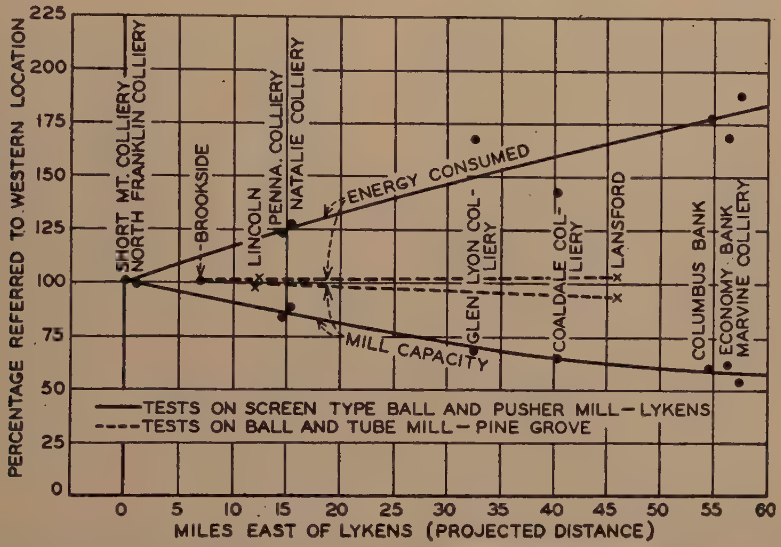

FIG 3.—RESULTS OF EXPERIMENTS WITH COAL FROM VARIOUS SECTIONS OF THE ANTHRACITE REGION.

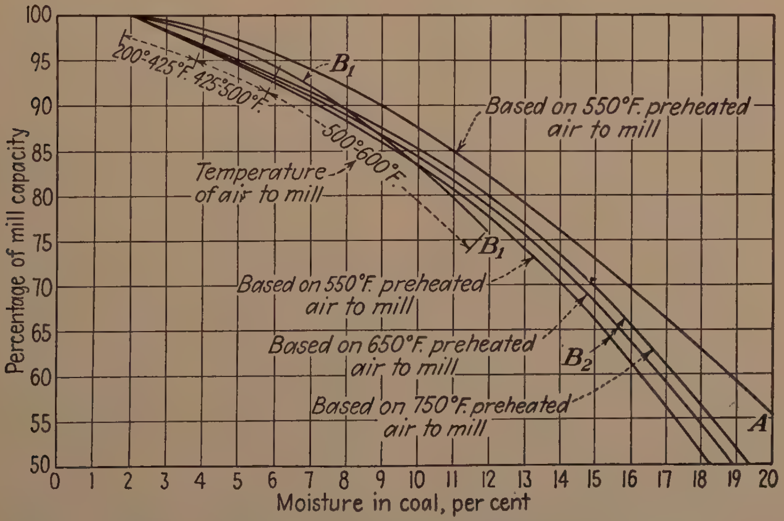

FIG 4.—EFFECT OF MOISTURE ON CAPACITY OF HORIZONTAL BALL AND TUBE MILL.
Curves from manufacturers' proposals: Curve A from manufacturer A; curves B_1 and B_2 from manufacturer B.

variation in the output of the vertical mill, depending on the hardness of the coal. Fig 3 shows data plotted on a geographical basis from west to east in the anthracite region.

In the ball and tube mill, the variation in capacity with location in the field and ash content is comparatively small. From meager data available, it appears that grindability varies from 60 Hardgrove in the western end of the region to 35 in the eastern. This also is illustrated in Fig 3.

The output of a given fineness with a given mill installation varies considerably with the initial moisture content of the coal entering the mill. It is not customary to use preheated air in the mill if external drying of the coal is done in advance, but when mill drying is used it is customary to use preheated air of about 500 to 550°F.

Capacity of the horizontal ball and tube mill drops considerably with an increase of moisture entering, as shown in Fig 4. As an example, in the winter the moisture content of the small anthracite may be at least 20 per cent, or even higher, as received. Even though preheated air is used in mill drying, the capacity is less than 60 per cent of what it would be with initial moisture less than 2 per cent, which means that for an installation to operate at full load on 100 per cent anthracite more mills would be required to take care of this wintertime condition than are needed when the coal is predried.

As mentioned, the moisture content of the small anthracite leaving the mines is inherently high because of the methods of preparation, and if the coal is not dried before it enters the mill, it will not be properly dried, even with preheated air in the mill.

GRINDING BALLS

The ball and tube mills usually are supplied with forged steel balls of 1¼ and 2-in. diameter, having a Brinell hardness of about 225 to 250. About 1941, this company began to use Ni-hard balls, having a Brinell hardness of 550 to 650. The balls are white cast iron alloyed with 4.5 per cent nickel and 1.5 per cent chromium, which is very beneficial in extending the life of the balls.

TABLE 2.—*Data on Balls*

Data	Standard Balls, 1/6/35 to 6/2/41	Alloy Cast-iron Balls, 6/5/41 to 6/30/42	Reduction in Cost Due to Use of Alloy Cast-iron Balls
Pine Grove			
Coal pulverized, net tons	627,465	106,785	
Ball consumption due to ball wear, lb	419,603	33,114	
Constant for ball wear, lb. per net ton	0.67	0.31	
Cost of balls, 6/17/42, per net ton balls	$130	$150	
Cost of balls at last quoted price per net ton coal pulverized	$0.044	$0.023	
Average cost of balls per net ton coal pulverized	$0.044	$0.023	$0.021
Annual ball cost based on 225,000 tons coal per year	$9,900	$5,200	$4,700
Hauto			
Annual ball cost, based on 250,000 tons coal per year	$11,000	$ 5,800	$5,200
Pine Grove and Hauto			
Total annual ball cost	$20,900	$11,000	$9,900

A test run on one of the horizontal ball and tube mills at the Pine Grove plant for 6½ years showed the loss in weight of the forged steel balls to be 0.67 lb per ton of coal ground. A test was run for one year using Ni-hard balls, which showed the loss in weight of balls to be only 0.31 lb per net ton of coal ground. These data are shown in Table 2.

Using the plain steel balls, the annual cost for ball consumption at this plant, where 225,000 tons of coal is ground per year, was $9900, whereas with the

Ni-hard balls the cost was only $5200, a reduction in this expense of $4700 per year. At the Hauto plant, where more than 250,000 tons of coal is ground per year, the saving in the annual expense of ball replacement is estimated to be at least $5200 owing to the use of Ni-hard balls. A test is now under way using copper-molybdenum steel balls.

These same facts apply to the use of serrated liners inside the mill casing.

On the vertical ball mill, the original parts generally were steel and the balls were made of chilled cast iron. However, it was found that the addition of a slight amount of nickel prolonged the ball life as well as the life of the various other parts of the mill subject to rapid wear.

Cost of Preparation

For the period of three years during which the vertical type of mill described was used exclusively at the Pine Grove plant and a wide variation in type of fuel was used, the maintenance cost was 41¢ per ton, including cost of parts and labor for installation, whereas with the horizontal ball and tube mill, the comparable cost was 8¢ per ton for the same items over a period of about 11 yr. The larger units installed at the Hauto plant show a cost of 8¢ per ton for a 2-yr period.

The finished product at Pine Grove has averaged 95 per cent through a 100-mesh sieve and 77 per cent through a 200-mesh sieve. At Hauto the corresponding figures are 98 per cent through a 100-mesh sieve and 85 per cent through a 200-mesh sieve.

The total operating and maintenance cost of preparing and pulverizing small-sized anthracite at this company's Pine Grove steam electric station has averaged 36¢ per ton for the 11-yr period mentioned and at the Hauto steam electric station, 42¢ per ton for a 2-yr period. This cost includes maintenance, labor and material, operating labor and drier fuel, but excludes cost of electrical energy. Use of electrical energy averaged 34.5 kw-hr per ton at Pine Grove and 36.5 per ton at Hauto for motors driving the drying, grinding and conveying equipment. Present-day costs of maintenance and total preparation are higher than the averages given previously, because of increased material and labor costs.

Conclusion

It should be noted that a public utility company must have equipment that will provide as nearly as possible continuous service, and for this reason there must be equipment that will dry the small anthracite during all seasons of the year and grind it without interruption in fuel supply to the boilers in sufficient quantity to run the boilers at full load. There are various ways of accomplishing this objective, and the two installations described herein have separately fired driers to drive off the moisture in the coal, so that the milling plant can and will operate on a constant basis the year round; also, even if there are minor interruptions in the mills themselves, there would still be an ample supply of coal on hand to operate boilers at full load without any interruption from this standpoint.

The new Sunbury plant, under construction, will depend on mill drying of anthracite, but for constant flow of coal to boilers when anthracite is extremely wet will use some bituminous coal with the anthracite.

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Increasing the Value of Coal Silts by Pelletization

By R. J. Day* and C. C. Wright,* Member AIME

(New York Meeting, February 1948)

Although data on the exact tonnage of recoverable coal silt are not known, the quantity produced in 1943 was estimated to be over five million tons for the anthracite region of Pennsylvania alone. Since that time the recovery of the No. 5 buckwheat size previously included in silt estimates has increased greatly thus reducing the total silt tonnage. As a compensating factor, however, the methods of recovering silt previously escaping into the rivers and streams have been markedly improved so that the net figure may not have changed materially. Comparable estimates for bituminous coal are lacking, but undoubtedly the silt represents a significant tonnage. With the trend toward increased wet cleaning of the smaller sizes of bituminous coal, it is to be expected that even greater quantities of silt will be available in the future.

The raw silt from anthracite collieries usually contains a high mineral matter content, but technological developments in fine coal cleaning have been making rapid advances and it appears probable that more extensive use of these methods will be employed in the near future. Bituminous coal silt in many instances contains a similar high mineral matter content. As with anthracite silt, however, the cleaning of this material, where necessary, appears both technologically and economically feasible.

With a view to upgrading the size and value of silt, considerable laboratory and pilot plant work has been conducted by a number of organizations on what is commonly known as pelletization. Actually the process is one of extrusion in which the moist silt is extruded from a die in the form of rods, which usually break into relatively short lengths—a few inches long—as they fall from the die. Fig 1 is a photograph of some extruded ⅜-in. diam pellets and of the anthracite silt from which they were produced

This technique of producing larger sized material from fines is by no means new, having been used extensively by the industrial carbon and clay industries for years.[1] The application of the process to coal silt appears, however, to be relatively new. The first major program of investigation dates back only to July 1945, when a group of engineers representing the Anthracite Institute, The Pennsylvania State College and the J. F. Pritchard and Co. conducted preliminary tests on the extrusion of anthracite silt at the Floridin Co. plant at Warren, Pa. Since that time these three organizations—individually, together, and in cooperation with other interested groups—have actively pursued work on various phases of the problem. Much of the early history and details of the laboratory and pilot plant work by the Anthracite Institute, has been reported by Mulcey and Eckerd.[2] It is the purpose of this paper to report additional information on the process, especially with respect to the factors influencing the quality and use properties of the product, and the current develop-

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[1] References are at the end of the paper.

ments relating to both anthracite and bituminous coal silt.

The Extrusion Machine

Three different extrusion machines have been used in the preparation of pellets. Two of these are laboratory size machines with capacities of less than one hundred to a few hundred pounds per hour, the third is a pilot plant machine with a capacity of 4 to 5 tons per hr depending upon the size of the pellets extruded. All three machines are similar in general construction and resemble a large meat grinder in appearance and operation. Fig 2 shows diagrammatically the essential features of the extrusion presses used in this work.

The material to be extruded is fed continuously from a hopper *A* to a feed screw *B* revolving in a cylindrical barrel *C* which delivers it to a revolving auger *D* in a grooved section of the cylindrical barrel. The auger forces the material through holes in the rigidly attached die plate *E* forming the feed into extruded rods or pellets. The machine illustrated also has a compaction ring *F* which serves to give the feed a preliminary compression before delivering it to a de-airing section of the screw feeder where, if desired, reduced pressure may be applied.

Detailed data on the extrusion press design characteristics such as clearance between auger and die face, ratio between die thickness and die hole (pellet) diameter, auger design, extrusion pressures and power consumption for different size pellets have been reported[2] for the laboratory scale

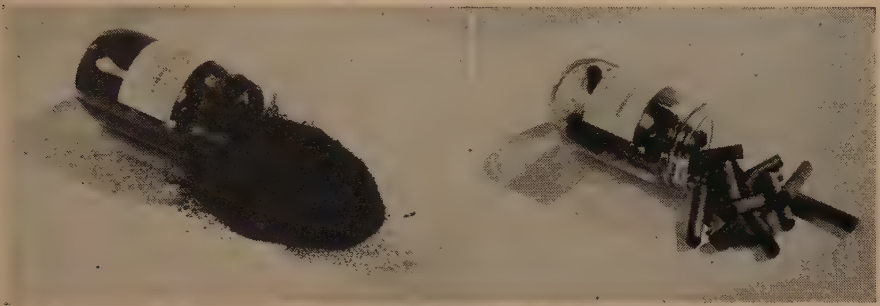

FIG 1—PHOTOGRAPH OF ⅜-IN. DIAM ANTHRACITE PELLETS AND SILT FROM WHICH THEY WERE PRODUCED.

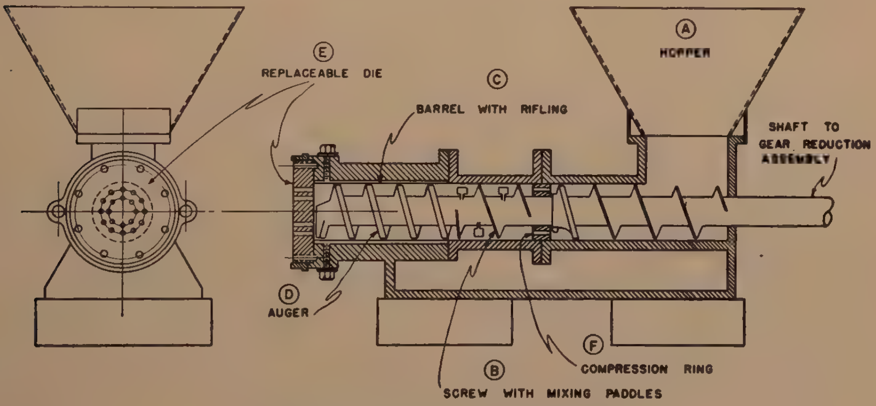

FIG 2—LABORATORY EXTRUSION MACHINE.

studies conducted by the authors and by the Anthracite Institute, and for the pilot plant studies by the Anthracite Institute. Little new information on these factors is currently available although in pilot plant tests on bituminous coal silt, the same general relationships have been found to apply as with anthracite silt. The data secured in the aforementioned tests had been used as the basis for the design of a new extrusion press by the J. F. Pritchard and Co. which, it is hoped, will overcome some of the mechanical difficulties experienced with the present pilot plant machine. This latter machine was not originally intended for coal extrusion work and for this reason several mechanical modifications were necessary before satisfactory coal extrusion could be secured.

THEORETICAL ASPECTS OF PELLET FORMATION

The mechanism by which an unconsolidated mixture of wet, fine sized coal can be converted into a coherent agglomerate by pressure alone may be similar in principle to that by which Kegel[3] has explained the binderless briquetting of lignite. The interstices between the particles form a family of capillaries of varying diameter, and any moisture present will arrange itself within the capillary network so that the strongest forces will be satisfied. Because in actual practice the particles are of varying size and shape, the mathematical treatment of such a system is extremely complex. For illustrative purposes, however, it is of interest to consider a relatively simple system consisting of uniformly sized spherical particles.

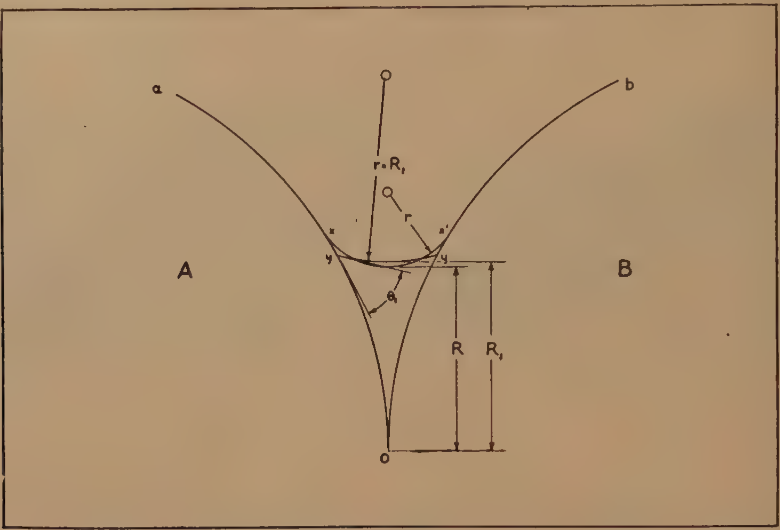

FIG 3—SEGMENT OF CAPILLARY FORMED BY CONTACT OF TWO SPHERES AT POINT *O*.

Consider first the capillary *a*, *o*, *b* formed by the contact of spheres *A* and *B* of Fig 3, and the influence of moisture content and size of the spheres upon the force holding the liquid in the capillary. For the special case where the spheres are completely wetted by the liquid (angle of contact $\theta = 0$) the force may be expressed[4] as a capillary pressure according to the general equation $P_c = \gamma \left(\frac{1}{r} + \frac{1}{R}\right)$

where P_c = capillary pressure in appropriate units (dynes per sq cm, lb per sq in., etc.)

γ = surface tension (dynes per cm, lb per in., etc.)

r = radius of curvature of the meniscus (cm, in., etc.)

R = distance between point of contact between the spheres and the meniscus (cm, in., etc.)

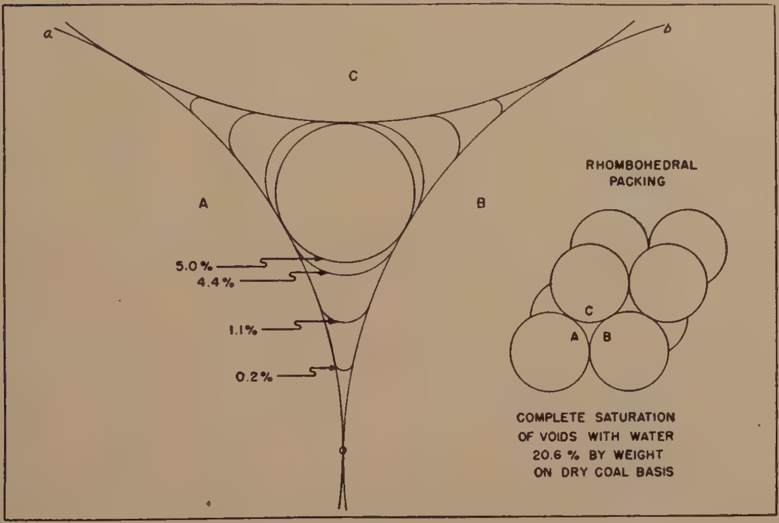
a
b
C
A
B
RHOMBOHEDRAL PACKING
5.0%
4.4%
1.1%
0.2%
C
A
B
COMPLETE SATURATION OF VOIDS WITH WATER 20.6 % BY WEIGHT ON DRY COAL BASIS

FIG 4—POSITION OF MENISCI AT DIFFERENT MOISTURE CONTENTS IN THE CAPILLARIES FORMED BY THE CONTACTS BETWEEN SPHERE A, B AND C.

If the centers of radii for r and R are on the same side of the interface, the signs are the same; if on opposite sides of the interface the signs are opposite. The convention is generally adopted that R is negative. For the coal-water-air system the equation may therefore be written $P_c = \gamma\left(\frac{1}{r} - \frac{1}{R}\right)$.

Assuming a rhombohedral system of packing, which is the closest configuration possible with uniformly sized spheres, the percentages of water present in a coal-water-air system have been calculated for the several positions of the meniscus shown in Fig 4. Using these data, and the capillary pressures calculated for particles of several different sizes, Fig 5 has been constructed. This plot shows graphically the relationships between particle size and moisture content, and capillary pressure at a single contact point.

For a system of spheres having a known packing configuration, the compaction strength can be calculated by considering the number of contacts and the magnitude, direction, and area, over which the capillary forces are acting. Such calculations are of little value to the present discussion, however, because in the pelletization of coal the particles are neither uniform in size nor shape so that calculated values of absolute compaction strength might even be misleading. It is of more significance to consider the factors that might influence the overall compaction strength in a system such as actually exists in the pelletization of silt.

From the data of Fig 5 it is apparent that particle size is of considerable importance. Not so readily apparent, however, is the fact that distribution of sizes is also a significant factor. If the particles are of uniform size they would have to be quite small before the forces could attain significant proportions. If, however, the distribution of sizes is such that the capillaries between the larger particles are successively filled with smaller particles it is readily apparent that a large number of very fine capillaries can be formed and that the capillary pressures, hence attractive forces, can attain significant proportions.

Because overall compaction strength is dependent upon the number of contacts between a given particle and those surrounding it, it follows that the closer the packing the greater the total force. It is not improbable that one of the primary functions of the extrusion machine in pelletization is that of compressing the particles into a configuration of maximum or closest packing.

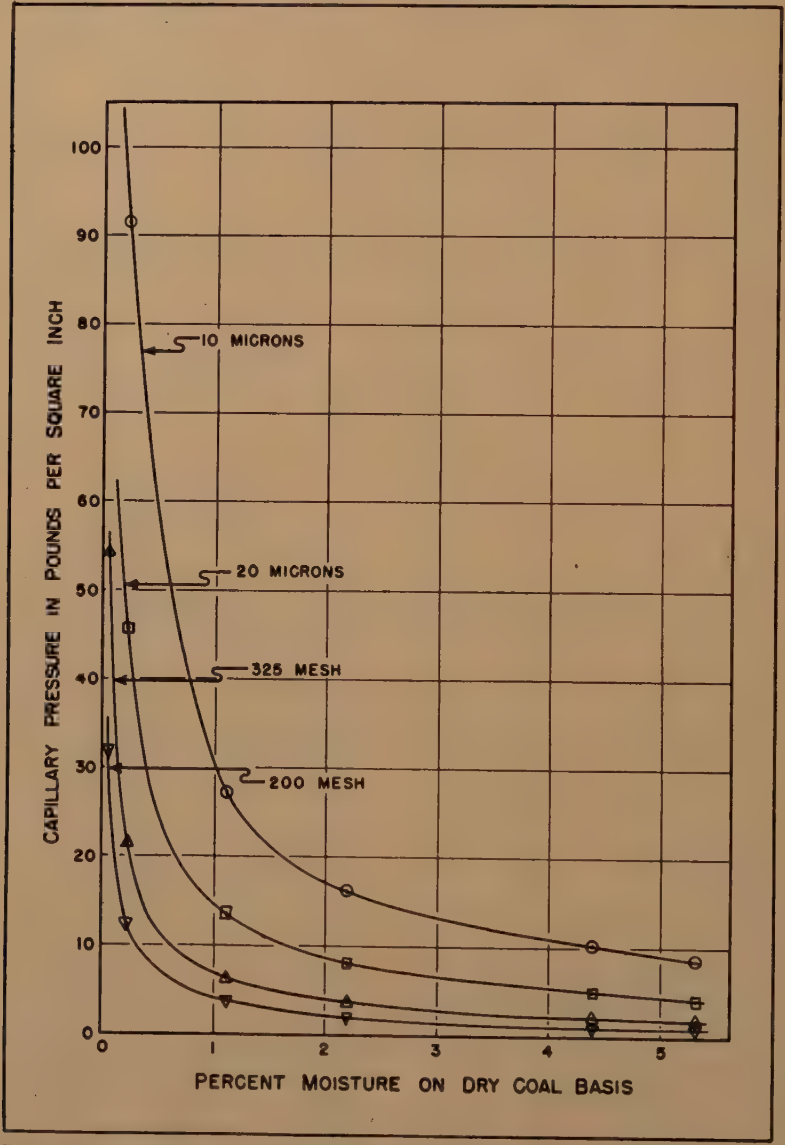

FIG 5—CAPILLARY PRESSURES AT VARYING MOISTURE CONTENTS FOR SPHERICAL PARTICLES OF CERTAIN SIZES WHEN IN CONFIGURATION OF RHOMBOHEDRAL PACKING.

The factor of particle shape is probably quite important. Flat, platelike particles, such as are encountered in coals, will furnish a potentially greater amount of contact surface between particles and, therefore, a greater cohesive force than between spherical particles. Such flat, angular particles will be more difficult to pack in the closest configurations. It is believed, however, that the molding action of the extrusion press and the pressures imposed will tend to crush and force these particles into a very close configuration.

From Fig 5, it may be seen that capillary pressure increases with decrease in moisture content. Obviously the area over which the capillary forces act decreases as the moisture content is reduced. The decrease in area is by no means as rapid as the increase in capillary forces, however, so that the overall effect of decreasing moisture content is that of increasing the compaction strength. During the extrusion process itself, moisture contents greater than that permissible when closest packing exists will either be squeezed out or will prevent maximum packing; moisture contents lower than the saturation value for closest packing may permit air pockets which, upon release of the extrusion pressure, will expand and disrupt the pellets.

In the basic equation, it is assumed that the particle surface is completely wetted by the liquid. If this condition of complete wetting does not exist, calculation of the capillary forces becomes exceedingly complex, yet for the coal-water-air system incomplete wetting is likely to exist. Fortunately the influence and importance of wettability can be demonstrated, without resort to a complicated mathematical treatment, as shown in the following illustration.

Consider the two spheres *A* and *B* of the coal-water-air system shown in Fig 3. When complete wetting of the coal surface exists, the angle of contact at x, x' is 0 degrees and the capillary pressure would be about 9 psi for particles of 10 micron diam. If, however, the moisture content remains the same but the coal surface is not completely wetted, the meniscus will assume a new shape. Taking as a specific example the case where the radius of the meniscus r is equal to R_1 then the capillary pressure $= \gamma\left(\frac{1}{r} - \frac{1}{R}\right) = 0$ and there will be no tendency for the two particles to be held together by capillary forces. In this particular example the angle of contact at the y, y' coal-water-air interface is approximately 45°. As shown by Brady and Gauger[5] such a contact angle is not improbable when dealing with a coal-water-air system. Wettability is, therefore, a very important factor in determining the capillary pressures. This suggests that the addition of a small quantity of a wetting agent might be highly beneficial in cases where incomplete wetting exists, since in addition to improving wettability it would also tend to increase slightly the surface tension of the liquid. Excessive additions of wetting agent would prove detrimental, however, as surface tension drops very rapidly after passing through a maximum at very low concentrations. Capillary pressure would be reduced in direct proportion to the reduction in surface tension as indicated by the basic equation.

EXTRUSION BEHAVIOR OF COAL SILTS

During the course of the extrusion studies, tests have been made using about a dozen different anthracite silts, several bituminous coal silts ranging in rank from low volatile to high volatile *A*, and one subbituminous coal silt. Studies have also been made on the extrusion of mixtures of anthracite and bituminous coal silts.

With the anthracite silts, a product of satisfactory strength characteristics was obtained in most cases and pellets were formed in all cases. From tests with a silt from which occluded clay materials have been carefully removed, the Anthracite Institute[2] concluded that the binding force was not associated with occluded clay content. However, the fact that no product of satisfactory strength was obtained from cleaned silts of relatively low ash content caused these investigators to suggest that possibly the inherent ash may impart high resistance to impact and abrasion to the pellets. A more probable explanation appears to be that presented in the preceding section. The lack of satisfactory pellets from cleaned silt is probably due to the fact that the cleaned product has a radically different size consist (distribution of sizes) from that of normal silts, and the distribution of sizes is not the optimum for maximum packing, hence for the formation of capillaries between particles sufficiently small to permit adequate capillary forces to develop. In certain cases, surface films of extraneous materials acquired during the cleaning operation, or surface oxidation during drying, may prevent satisfactory wetting and cohesion of particles. Brady and Gauger[5] have shown that wettability of a coal surface with water is dependent upon the previous history of that surface.

The response of bituminous coal silts to extrusion has been somewhat more erratic than that of anthracite silts. In fact, some bituminous coal silts did not respond satisfactorily under any of the conditions investigated, and caused plugging and stalling of the machine. Such coals would not flow freely through the die, but instead formed a dense plug at the die face. Increase in moisture content of the feed to improve the fluidity of the material, failed to improve extrusion behavior because the excess water was squeezed out of the coal and ran out of the bottom of the die while the coal in the upper part formed a relatively dry cake. This behavior suggests the possibility that their surface properties were such that the particles were not wetted by water. A more thorough study of this problem, particularly along the lines of modifying the size consist and the addition of wetting agents to insure wetting of the coal surfaces, may result in the production of satisfactory pellets even from these seemingly unsuitable silts.

A limited number of mixtures of anthracite and bituminous coal silts have been investigated and no difficulty experienced in producing satisfactory pellets in cases where the bituminous coal silt itself produced a satisfactory product. Even when using bituminous silts, which alone failed to extrude satisfactorily, mixtures containing 20 pct or more of anthracite silt produced satisfactory products.

Of the two dozen or so natural silts thus far studied, only two have failed to produce pellets; a few others have yielded pellets considered of suitable strength only after several passes through the machine (change of size consist) or when binders or lubricants were added.

FACTORS INFLUENCING THE EXTRUSION PRODUCT

Other than design characteristics of the extrusion press, a number of factors influence markedly the quality of the extruded pellets. Among these, the following have been investigated: moisture content of feed; mixing; size consist of feed; addition of binder or lubricants; addition of waterproofing agents; rate of drying; and carbonization of product. Each of these items is discussed separately in the following paragraphs.

Moisture

As would be expected from theoretical considerations, moisture content of the feed had a very marked influence upon the quality of the resulting product. If the moisture content was too low, the cohesion of particles was not obtained and the product was weak and crumbly; if too high the product was weak and mushy. A good product emerging from the press was firm, but could be crushed readily between the fingers and would not withstand deep piling or rough handling. Upon drying, however, the strength increased, and when the moisture content had been reduced to about 10 pct the product could be safely handled. Further drying increased the strength materially.

The data of Table 1 for one anthracite silt indicate the range in moisture content of feed and product over which a satisfactory product was obtained from the laboratory press for various sizes of extruded pellets. These data are of value only as indicating the range of moisture contents, because it has been found that the absolute values vary slightly with the design of the particular extrusion press. These results are in general agreement with those of Fritsche[6] who reported an optimum moisture content of about 16 pct for the pressure briquetting of lignite without a binder.

TABLE 1—*Moisture Content of Feed and Extruded Product*

Pellet Diameter (In.)	Pct Moisture Content (Dry Coal Basis)	
	Feed	Product
⅛	17 to 22	15.5 to 19
⅜	17 to 21	15 to 18.5
⅝*	22 to 26	22 to 24
¾*	23 to 26	22 to 23

* The laboratory press is not designed for the production of pellets over ½-in. in diam and in order to produce a satisfactory product more water than normally required was used. The pellets were softer and more moist but hardened satisfactorily on drying.

In the pilot plant machine, anthracite silts with moisture contents ranging from 13 to 21 pct have been extruded satisfactorily in ¼-, ⅜-, and ½-in. diam sizes. Similarly, bituminous silt of 15 to 20 pct moisture content has been extruded to produce satisfactory ½-, ¾- and 1-in. pellets.

Mixing and De-airing

In laboratory work, it was found advantageous to pass the material through the press two or more times in order to obtain thorough mixing and thus produce an improved product. The same improvement could usually be accomplished by mixing the coal with the required amount of moisture and then allowing the mixture to age for a few hours before extruding. A similar phenomena has been observed[7] in the ceramics industry where an improved product results when the clays are aged before molding or pressing. In this way the moisture is allowed to penetrate the coal mass, wetting the surface and removing the adsorbed air film. A method used in the ceramics industry for removing this air film involves the application of vacuum during extrusion, with resultant improvement of the product. It has been reported[8] that the additional power expended to operate the vacuum pumps is largely saved by improved performance of the extrusion press. The application of vacuum de-airing during extrusion of coal pellets may alleviate some of the operating difficulties, reduce the number of passes necessary and improve the resulting product.

Size Consist and Pulverization

A number of tests have been made to determine whether the initial size limits of the silt are critical. In general, the results suggest that within the limits encountered in typical silts, top size about ⅛-in., the initial size is not critical. Satisfactory pellets have been made from silts ranging in top size from 27 pct on 14-mesh to 100 pct through 200-mesh. The latter was not a natural silt, but an anthracite silt ground to this size in a hammer mill.

Although within limits, top size appears not to be of major significance, size consist (distribution of sizes) appears to be of considerable importance. Frequently the size consist of a silt will not be that most satisfactory for extrusion. In part, the extrusion process itself will take care of this deficiency by grinding the material to a new size consist. Over the limited range tested, it appears that most of the grinding action occurs during the first pass through the press, but in cases where the consist differs radically from the optimum, additional passes will modify the consist further and will usually improve the quality of the product. It has been observed also that for the two sizes of pellets studied most extensively, ⅛- and ⅜-in., the size consist is modified to the same extent despite the fact that more power is required for the extrusion of the smaller size and the opportunity for grinding appears to be greater. This suggests that the silt is ground to a consist that will give maximum packing in each case and that little additional change occurs after this condition has been attained.

TABLE 2—*Size Consist of Silt and of Material in Extruded Pellets*

Material	Screen Analysis (Tyler Series) Per Cent							
	6 × 14	14 × 20	20 × 35	35 × 65	65 × 100	100 × 150	150 × 200	−200
Anthracite Silt (feed material)	3.6	6.4	25.4	29.0	12.0	6.6	5.0	12.0
⅛-in. Pellets (Lab Machine)	0.6	0.7	12.9	19.7	14.6	6.9	8.7	35.9
⅜-in. Pellets (Lab Machine)	0.6	0.8	12.0	22.0	13.0	7.0	9.6	35.0
Bituminous Silt (feed material)	2.3	4.1	28.5	35.0	14.5	6.4	3.8	5.4
⅜-in. Pellets (Lab Machine)	0.2	1.0	9.2	14.9	9.0	5.8	5.4	54.5
Subbituminous Silt (feed material)	7.5	19.5	33.2	15.6	6.8	4.0	3.3	10.1
⅜-in. Pellets (Lab Machine)	2.4	9.2	26.5	15.6	5.6	2.3	1.8	36.6
Anthracite Silt								
¼-in. Pellets (Pilot Machine)	0.5	3.0	13.1	20.1	10.3	6.0	5.4	41.6
½-in. Pellets (Pilot Machine)	0.6	3.2	13.0	20.4	10.5	6.2	6.0	40.1
Bituminous Silt								
1-in. Pellets (Pilot Machine)	0.1	0.2	1.6	16.5	20.0	12.5	10.5	38.6

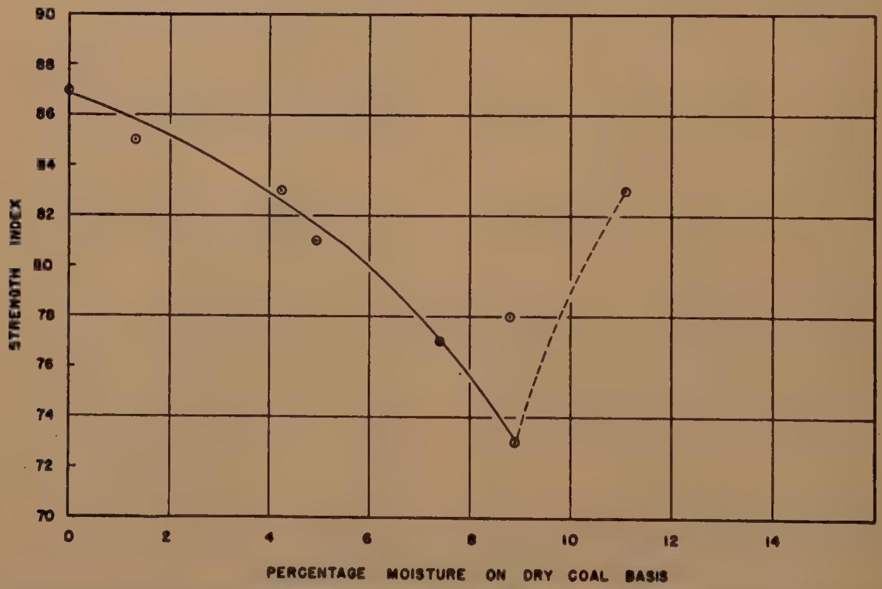

FIG 6—CHANGE IN STRENGTH INDEX FOR ⅜-IN. ANTHRACITE PELLETS WITH CHANGE IN MOISTURE CONTENT.

Results of the screen analyses of the original silt and of the material in the pellets after extrusion from the laboratory press are shown in Table 2 for ⅛- and ⅜-in. anthracite pellets from the same silt, for ⅜-in. bituminous pellets from a bituminous silt, and for ⅜-in. pellets from a subbituminous silt. Screen analyses for both anthracite and bituminous pellets from the pilot plant operations are also shown. The screen analyses of the pellets were obtained by allowing the pellets to disintegrate in a beaker of water, and wet screening the resultant material.

Drying

When obtaining a satisfactory extruded product, the pellets usually emerge from the press at temperatures in excess of 130°F and with moisture contents of 13 to 20 pct, depending upon the initial moisture content of the feed, the die thickness, and the diameter of the pellets. In this condition the pellets are relatively weak and will not withstand extensive handling or deep piling without deformation. Some form of drying is therefore essential.

Laboratory tests indicate that after drying to a moisture content of about 10 or 11 pct, the product will withstand normal handling and piling, but that the resistance to crushing and abrasion is still relatively low. The increase in mechanical stability of the pellets, as a result of moisture content reduction, is shown in Fig 6 for a sample of ⅜-in. pellets prepared from an anthracite silt. These results suggest that in the range from about 7 to 0 pct moisture content the strength index is almost inversely proportional to moisture content.

TABLE 3—*Influence of Rate of Drying upon Pellet Strength*

Pellet Size In.	Temp. of Pre-drying* °F	Air Circulation Rate through Bed in $Ft.^3$ per Min. per Ft^2 of Bed Area	Time of Pre-drying*	Strength Index†
⅜	70	0	6 hr	91
⅜	70	50	90 min.	87
⅜	70	100	65 min.	80
⅜	160	50	17 min.	84
⅛	70	0	4 hr	90
⅛	160	50	8 min.	83

* One-inch layer of pellets pre-dried to 6.5 pct moisture content followed by oven drying at 210°F.
† Oven dried at 210°F.

Laboratory tests also suggest that the rate and temperature of drying may have an influence upon the strength index of the final product. The data of Table 3 suggest that as the rate of drying is increased, the strength index of the product decreases. This same result has been observed in the drying of pelletized catalyst.[9]

Waterproofing

The extruded pellets after drying possess the undesirable property of disintegrating upon direct contact with water. For certain uses or where indoor storage could be used this would not be an important factor, but where outdoor storage is desired, the product would have to be weatherproofed. With anthracite and bituminous coal pellets there are several ways in which this can be accomplished, such as the addition of various binders or a mild low temperature carbonization.

Of the many binders investigated, one of the most promising appears to be low cost tar emulsions* from heavy oil water-gas operations. From 3 to 5 pct of the tar emulsion, containing approximately 70 pct tar, added to the silt prior to extrusion resulted in complete waterproofing. In some instances the tar also improved the ease of production and the strength of the product, but insufficient data have been secured to generalize upon these latter advantages.

Similar results were obtained using various wax emulsions which like the tar emulsion break upon drying and thus form a water repellent film around each particle. None of the waxes tried appeared to be more efficient than the tar, however, and as their cost was relatively greater, the use of tar emulsion appears to offer more promise.

Certain of the pelletized bituminous coals after extrusion appear to have a mild degree of water repellency. This is not a general characteristic of all bituminous coal pellets and must be attributable to some inherent property of these particular coals or to the conditions of extrusion. This phenomenon has not yet been investigated thoroughly.

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* Separator Tar. Sp. Gr. 1.16; Engler Visc. 50 cc at 60°C 483 sec.

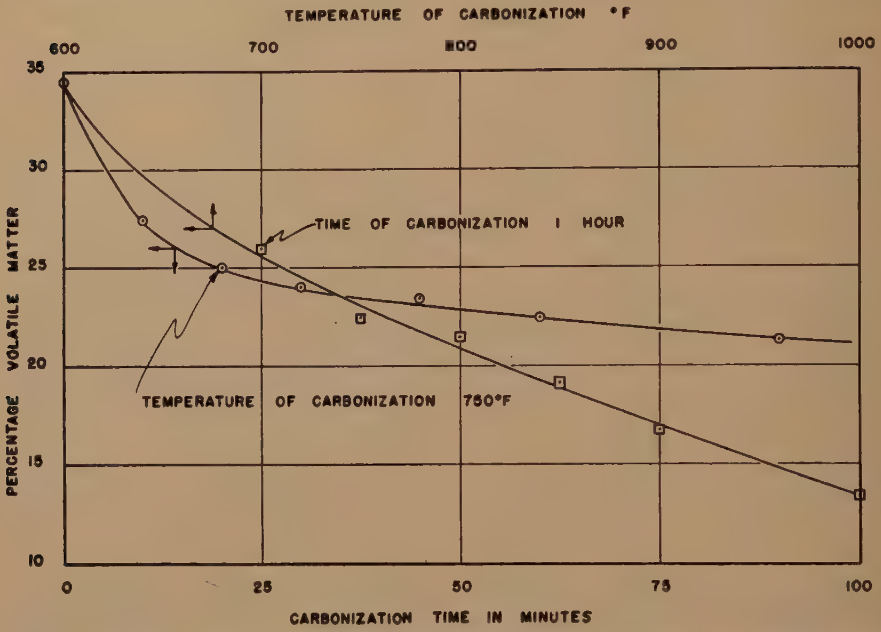

FIG 7—CHANGE IN VOLATILE CONTENT OF BITUMINOUS COAL PELLETS AS A RESULT OF LOW TEMPERATURE CARBONIZATION AT CONSTANT TEMPERATURE FOR VARYING TIMES AND CONSTANT TIME FOR VARYING TEMPERATURES.

As pointed out by Gauger[10] low rank coals have better wettability toward water than do the higher rank coals, but the wettability toward organic binders such as tar and pitch is poorer. It would be expected therefore that the waterproofing of low rank coal with the tar or wax emulsions would be less satisfactory than for anthracite and bituminous coals. This proved to be the case. Additions of up to 5 pct of the organic waterproofing agents to the subbituminous silt failed to produce satisfactory water repellency and served only to decrease the strength of the pellets produced.

A mild carbonization at 750°F of bituminous coal pellets and pellets made from mixtures of bituminous coal and anthracite resulted in a waterproof product and, as discussed in the following section, in an increase in the mechanical strength of the product. As would be expected, however, the carbonization of pellets made from subbituminous coal did not result in any improvement in water repellency or strength properties.

Carbonization

Extruded pellets from a number of bituminous coals, anthracite-bituminous coal mixtures, and a subbituminous coal, have been carbonized on a laboratory scale at 750°F to determine the suitability of such a product for use as a smokeless fuel. The response to this low temperature carbonization was quite varied. Some of the bituminous coal pellets swelled to two or three times their normal size as might be expected, while others showed little or no swelling even though coals from the same source are known to have fairly high swelling properties. The behavior of the pellets from anthracite-bituminous coal mixtures showed no significant swelling although carbonizing satisfactorily, and the pellets made from the subbituminous coal showed no carbonization effect except loss of volatile matter. With the exception of the subbituminous pellets, all showed an improvement in strength properties, although the degree of improvement varied considerably.

The reduction in volatile matter of a sample of ⅜-in. pellets from a high volatile bituminous coal as a function of time of carbonization at 750°F and as a function of temperature for a fixed time interval of one hour was determined and the results are shown in Fig 7. These quantitative results are specific only for the particular carbonizing conditions employed in the test, but are indicative of the general trends. Test data on the strength of carbonized and uncarbonized pellets from the same batch appear in Table 4.

TABLE 4—*Strength Index of Pellets and Standard Fuels*

Material	Source	Vol. Matter*	Binder	Pellet Diam (In.)	Extrusion Press	Strength† Index
Anthracite Silt No. 1	N. Field	9.0	0	⅛	Lab	90
Anthracite Silt No. 1	N. Field	9.0	0	⅜	Lab	91
Anthracite Silt No. 1	N. Field	9.0	0	⅝	Lab	91
Anthracite Silt No. 1	N. Field	9.0	0	¾	Lab	84
Anthracite Silt No. 2	N. Field		0	¼	Pilot	81
Anthracite Silt No. 2	N. Field		0	½	Pilot	79
Anthrafines No. 1	S. Field	4.6	0	⅛	Lab	36
Anthrafines No. 1 (−65 mesh)	S. Field		0	¼	Lab	48
Anthrafines No. 1 (−200 mesh)	S. Field		0	⅜	Lab	89
Anthrafines No. 1	S. Field	4.6	5 pct Floigel	⅜	Lab	78
Anthrafines No. 1	S. Field	4.6	6 pct Lignone	⅜	Lab	92
Anthrafines No. 1	S. Field	4.6	3 pct *F* + 3.7 pct *L*	⅜	Lab	91
Anthrafines No. 1	S. Field	4.6	5 Pct *F* + 6 Pct *L*	⅜	Lab	95
Bit. Slurry *A*	Ohio No. 8	41.4	0	⅜	Lab	93
Bit. Slurry *A*	Ohio No. 8	41.4	Carbonized‡	⅜	Lab	95
Bit. Silt *B*	Ohio No. 8	41.5	0	⅜	Lab	58
Bit. Silt *B*	Ohio No. 8	41.5	Carbonized‡	⅜	Lab	79
Bit. Silt *C*	Ohio No. 8	42.8	0	⅜	Lab	71
Bit. Silt *C*	Ohio No. 8		Carbonized‡	⅜	Lab	92
Bit. Cyclone Dust	Upper Freeport	33.0	0	⅜	Lab	0
Bit. Silt *D*	Fulton and Barnett		0	1	Pilot	85
Subbit. Silt	Drumheller (Alberta)	42.2	0	⅜	Lab	94
25 pct Bit. Cyclone Dust + 75 pct Anth. Silt No. 1			0	⅜	Lab	74
25 pct Bit. Cyclone Dust + 75 pct Anth. Silt No. 1			Carbonized‡	⅜	Lab	85
50 pct Bit. Cyclone Dust + 50 pct Anth. Silt No. 1			0	⅜	Lab	68
80 pct Bit. Cyclone Dust + 20 pct Anth. Silt No. 1			0	⅜	Lab	64
60 pct Bit. Silt A + 40 pct Anth. Silt No. 1			0	⅜	Lab	76
80 pct Bit. Silt A + 20 pct Anth. Silt No. 1			0	⅜	Lab	90
Treverton Anthracite No. 1 Buck	S. Field	10.2				97.5
Tamaqua Anthracite No. 2 Buck	S. Field	4.5				99.5
Blast Furnace Coke ½ × ¼ in.						99.5
Bituminous Coal ½ in. × 16 mesh	Lower Kittanning	28.0				82.5
Bituminous Coal ½ in. × 16 mesh	Pittsburgh	40.0				96.5

* Ash and moisture free basis.
† Oven dry samples.
‡ Carbonized for one hour at 750°F.

Lubrication

An interesting phenomenon was observed during the use of wax emulsions for the purpose of weatherproofing the pellets. It was observed that as little as one-half per cent of wax acted as a lubricant to such an extent that the power consumption was reduced to about one-half that required for the nonlubricated extrusion. Coals that had previously given trouble when fed too rapidly to the press could be handled readily when the wax emulsion was used and the through-put thereby increased. This behavior appears to be similar to that encountered in the clay working industries where lubricants in the form of oils, steam and even electricity—in which the phenomenon of electro-osmosis is utilized—have been successfully employed.[11] Whether the additional cost of the lubricant can be justified in terms of decreased power consumption and increased through-put cannot be definitely established until pilot plant data have been obtained upon which more accurate cost estimates can be based.

PROPERTIES OF EXTRUDED PELLETS

The properties of the extruded pellets, especially as they influence the possible use of the product, are of interest. These properties and probable behavior as indicated by laboratory tests have been evaluated and are discussed individually in the following sections.

Strength and Handling Properties

A property of major interest in any consideration of the extrusion process is the strength of the product and its ability to withstand normal handling. Since in general the quantity of material available for test as a result of laboratory extrusion studies was insufficient for the regular A.S.T.M. shatter and tumbler tests, a modification of theTy-Lab Tester Method[12] previously used in assaying the relative strength of laboratory blend cokes was used.* A somewhat different method was subsequently adopted by the Anthracite Institute[2] which is more suited to routine testing of large numbers of samples. The two tests appear to give values that are numerically similar.

The strength index, Ty-Lab method, of a few representative samples of pellets from anthracite and bituminous silts and from anthracite-bituminous silt mixtures are shown in Table 4 together with data on the strength of a number of standard fuels. In general a strength index of 80 or better is considered to indicate good handling properties,* although pellets with a strength index below this value can undoubtedly be used successfully where the conditions of use do not promote severe degradation.

Physical Properties

Bulk densities, pellet densities, porosities and apparent densities of the pellets are shown in Table 5.

TABLE 5—*Physical Properties of Pellets*

Size Inches	Material	Bulk Density Lbs per Ft^3	Pellet Density G per Cc	Porosity Per Cent	Apparent Density of Solids G per Cc
1/8	Anthracite	43.0	1.30	17.5	1.58
3/8	Anthracite	47.5	1.32	18.0	1.61
5/8	Anthracite	46.5	1.27	19.6	1.58
3/4	Anthracite	45.5	1.22	23.3	1.59
3/8	Bituminous	37.0	1.12	19.4	1.39

Since the pellets could be cut and filed to any convenient length the volumes were easily measured. The pellet density was then obtained by dividing the weight by the volume. The porosity was measured by saturating the voids in the pellets with a special low vapor pressure oil under vacuum. The pellets did not disintegrate in the oil which was easy to handle because of its low viscosity, low volatility and ready wettability.

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* This method is as follows: 50 g of pellets are placed in a wide mouth quart jar (Fisher Scientific Co. 3-315) with screw top. After being padded to prevent shifting and breakage, the jar is placed on its side in a Ty-Lab Tester pan and the lid clamped securely in place. The jar and contents are shaken vigorously for 5 min. by the up and down motion of the Ty-Lab Tester. After shaking, the contents of the jar are removed and screened. The percentage of +16-mesh remaining (all original material was +16-mesh) is reported as the strength index. In order to insure comparable results, all samples are oven dried at 210°F prior to strength test. Duplicate tests are performed and if the results do not agree within ±1 pct, the test is repeated.

* The Anthracite Institute[2] tests have shown that material with a strength index of 80, by their test method, withstood severe handling without excessive degradation.

Resistance to Gas Flow

Resistance to gas flow of beds of various sizes of pellets has been determined in the standard apparatus used for that purpose and described by Barclay.[13] These tests showed that the resistance to gas flow of ⅛-in. pellets falls between the values for No. 1 buckwheat (5/16 × 9/16 in.) and pea (13/16 × 9/16 in.) sized anthracite. The resistance of the ⅜-in. pellets falls between the values for pea and chestnut (1⅝ × 13/16 in.) sized anthracite and the resistance of ⅝-in. pellets is almost equal to that of chestnut size anthracite.

It is apparent from these results that the pellets offer appreciably less resistance to gas flow than do the prepared sizes of anthracite of comparable sizes. This is due to the rod-like shape of the pellets which tends to form a more permeable bed. This is advantageous in any type of fuel burning equipment because the same quantity of air can be used with a smaller driving force whether it be forced, induced or natural draft.

Combustion Behavior

Tests on the combustion characteristics of the pellets have been made in the apparatus used at the College for studies on the fundamentals of combustion. This unit consists of a stainless steel retort in which a single layer of the fuel, supported on a stainless steel grate, is burned under controlled conditions of temperature and air supply. The loss in weight as a result of burning is determined at frequent intervals by means of a torsion balance from which the grate and burning charge are suspended. For the tests herein reported the retort temperature was maintained constant at 1870°F. The grate and fuel were introduced into the hot furnace and the charge permitted to attain its own temperature level. After the first few minutes its temperature fell to that of the furnace. A fixed percentage of excess air was supplied at all times during the burning and was the same for all tests.

Test data for two sizes of anthracite pellets, one size of bituminous pellets and

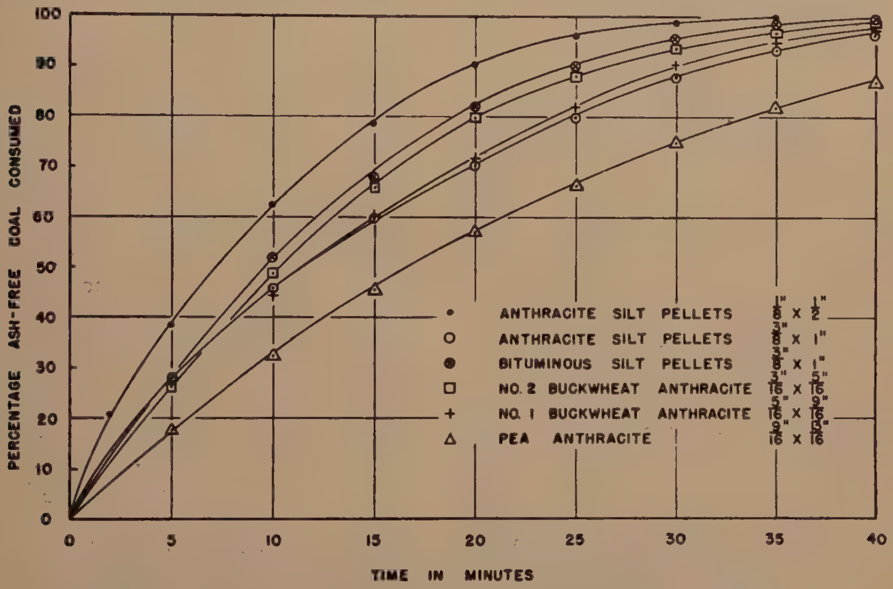

FIG 8—COMPARATIVE BURNING RATES OF PELLETS AND SIZED ANTHRACITE.

for three sizes of natural anthracite, are shown in Fig 8. From these data it may be concluded that the "apparent reactivity" (per cent of ash-moisture free coal consumed per unit of time) is higher for the pellets than for natural anthracite of comparable size; this, despite the fact that the pellets contain appreciably higher ash contents. Thus the pellets which have a lower fuel bed resistance than natural anthracite of comparable size have a higher "apparent reactivity." This combination of characteristics should be advantageous in any combustion or gasification applications.

The physical behavior of the anthracite pellets during combustion is also of interest. They maintained their shape, even when cold pellets were charged into the furnace at 1870°F, with no signs of decrepitation or spalling as a result of the thermal shock. The ash resulting from the pellet retains the original shape of the pellets while shrinking to about ⅔ to ½ the original size. The ash appears to fuse at points of contact forming a very porous semi-sintered mass with strength almost as great as that of the original pellets. It is possible that pellets from silts having markedly different ash fusion characteristics will behave differently, but all the anthracite pellets thus far studied in the laboratory tests exhibited this property.

The bituminous pellets, as would be expected, varied in their physical behavior with the origin of the coal. No decrepitation or spalling was observed with any of these pellets, although some had a tendency to fuse and agglomerate and a few exhibited strong swelling characteristics. Pellets from a High Volatile A coal of 41.4 pct volatile matter content, however, showed no appreciable swelling or agglomerating tendencies and burned freely. Unlike the ash of the anthracite pellets tested, however, the ash from the bituminous pellets studied had no appreciable strength and broke away from the pellets as a fine ash upon being disturbed.

Behavior Under Gasification Conditions

In view of the high "apparent reactivity" and low fuel bed resistance, the suitability of pellets for gasification purposes has been considered. To be satisfactory for this application the ignited pellets must be able to withstand the pressure of about 400 psf exerted by the weight of fuel above the high temperature zone, must maintain their strength in the presence of steam, and should produce no more fly material at the gas velocities encountered than do the sized fuels normally employed. To determine these characteristics the following tests were performed. Only anthracite pellets have been investigated up to the present.

To test the strength of the pellets when burning in a fuel bed under an imposed weight load, the apparatus shown in Fig 9 was devised. The pellets are placed on a 6-in. diam stainless steel grate while a perforated stainless steel plate 4½-in. in diam rests on the top of the fuel bed. After the fuel is ignited and is burning briskly, weights are added to the platform to impose a simulated load. A lever indicating arm attached to the platform as a fulcrum indicates any movement of the plate resting on top of the fuel bed.

In the tests, no indication of sudden or serious breakdown of the pellets was observed, the depression of the bed being proportional to the weight added. The depression indicating apparatus itself exerted a pressure equivalent to 95 psf and in one series of tests this load was maintained for 10 min. before adding additional weights in order to determine the normal settling due to the burning alone. Over the following 2-min. period, weights equivalent to a load of 600 psf were added while vigorous burning was continued. A slight (2 to 3 pct) compression of the bed resulted with all three sizes of pellets tested and with the rice size anthracite run as a control. Other than this compression, the normal rate of depression as a function of burning was maintained.

Tests under a constant load of 600 psf were also made under gasification conditions with an air-steam ratio of 2 to 1. Results are shown in Fig 10 in comparison with rice anthracite and pellets burned with air only at the same load conditions, and with pellets burned with air only under virtually no applied pressure. These results show that the presence of the steam has no significant effect. They also show that the depression of the bed is substantially less for the pellets than for the rice anthracite. This illustrates clearly the relative strength of the pellet ash as compared to the anthracite ash. Fig 11 shows photographs of the ash beds resulting from depression tests on ⅛ and ⅜-in. pellets and on rice anthracite burned under a maximum applied load of 900 psf. The bottom view pictures appear larger than the top view because in removing the ash bed from the grate to the pan in which it was photographed the ash spread out over a larger area.

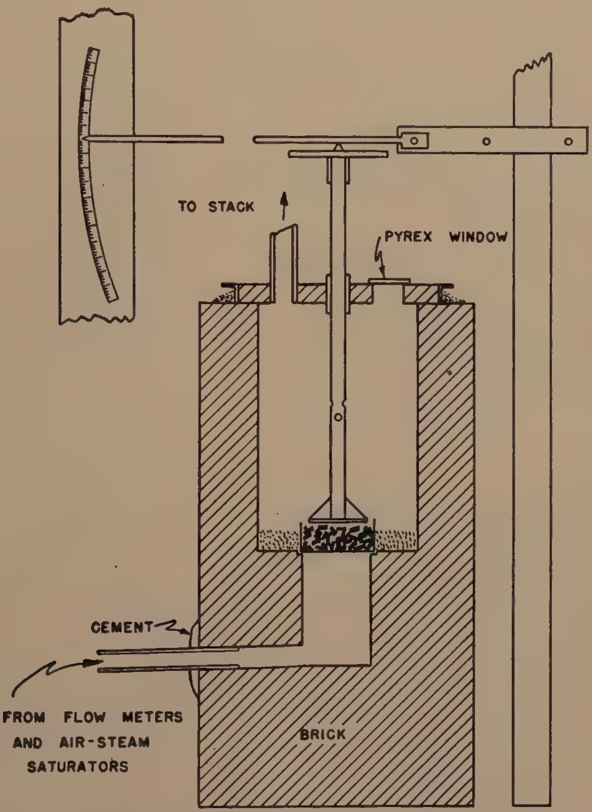

FIG 9—APPARATUS FOR DETERMINING THE DEFORMATION OF BURNING PELLETS UNDER IMPOSED LOADS.

Tests were also performed to determine the fly material resulting from the gasification of pellets and rice size anthracite when gasified at a rate of 20 lb of carbon per sq ft of grate area per hr—a rate comparable to typical producer gas operations—in a laboratory producer. The fly material was collected in a large cyclone which was backed up by a glass wool filter. With the ⅜-in. pellets, the total fly material collected was 0.003 pct and with rice size anthracite 0.005 pct. These results suggest that the pellets are at least as satisfactory as, if not more so than, sized anthracite.

Pilot Plant Operations

Two series of pilot plant scale test operations have been conducted within the past year during which several hundred tons of silt have been successfully extruded to produce pelletized fuel. The first series of tests conducted cooperatively by the Anthracite Institute, The Glen Alden Coal Co. and the J. F. Pritchard and Co. was made using a variety of anthracite silts, and the second series by the Rockhill Coal Co. and the J. F. Pritchard and Co. using ⅛-in. low volatile bituminous coal and bituminous coal silt. Fig 12 shows a close up view of the pilot plant machine extruding ¾-in. bituminous coal pellets.

The capacity of the machine, nominally rated at 4 tons per hr, varies with the size of pellets extruded, being somewhat below rating for ⅛-in. pellets, and exceeding rating by more than 25 pct when the pellet size is increased to 1-in. in diam. The power consumption also varies with the size of the pellets, being generally lower for the larger sizes. Depending upon the size of pellets and the particular material extruded, power consumption has varied from about 12 to 20 kw-hr per ton of material extruded. Data on the extrusion pressure with the pilot machine have not been obtained, but comparing the power consumption and texture of the product from laboratory and pilot plant operations on the same silts it is estimated that the pressures are in the range of 300 to 600 psi depending upon the size of the extrusion product.

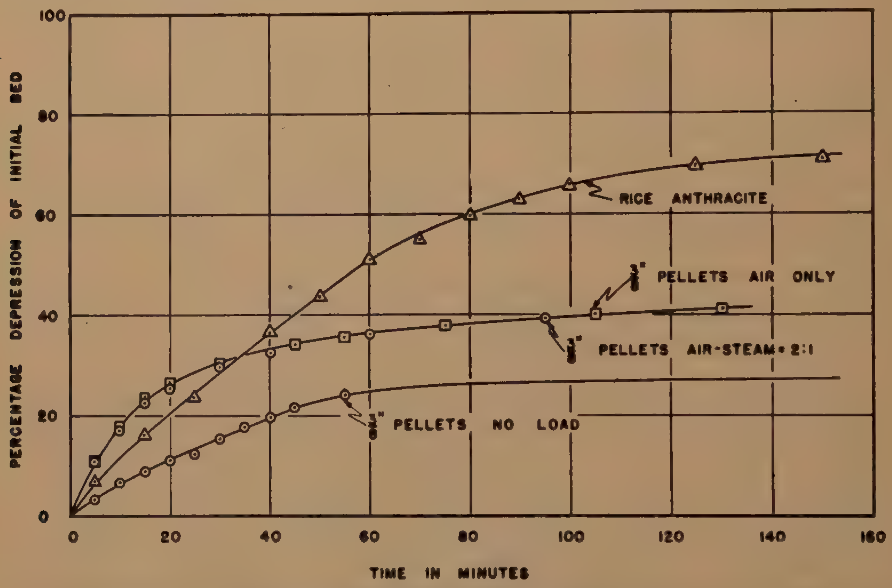

FIG 10—DEPRESSION OF FUEL BED UNDER IMPOSED LOAD EQUIVALENT TO 600 PSF DURING THE BURNING PELLETS AND SIZED ANTHRACITE.

Wear on the auger and die of the pilot machine has been surprisingly low, although sufficient tonnages have not yet been extruded to justify detailed cost estimates. Many of the early defects in the design of the machine, the dies, and the augers, were rectified during the anthracite tests, and have not been encountered during the pilot plant tests using bituminous coal silt. During the early tests with anthracite silt auger wear appeared to be excessive, but this was later shown to be due to design and materials of construction. Even when excessive wear occurs at the leading edge of the auger flights, repair is relatively simple and inexpensive.

The product, resulting from the pilot plant extrusion operations has been tested on a variety of commercial combustion

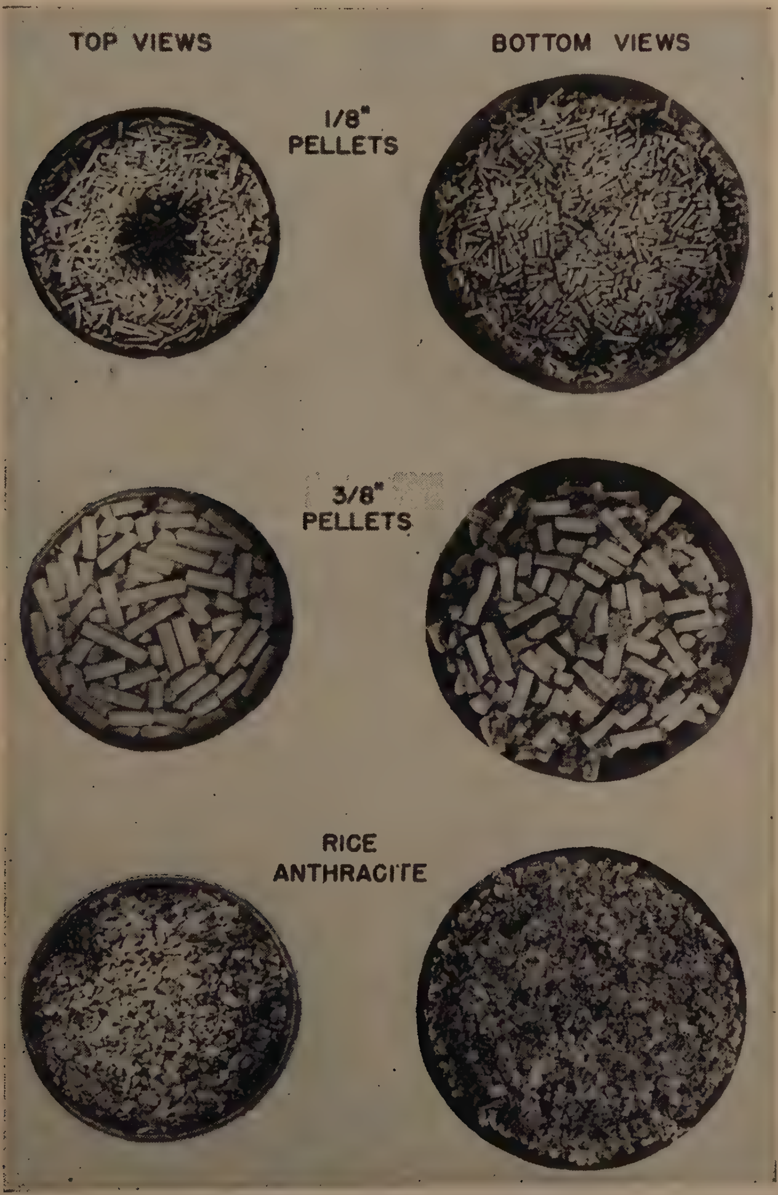

FIG 11—TOP AND BOTTOM VIEWS SHOWING CONDITION OF PELLET AND COAL BEDS AFTER STRENGTH TESTS MAXIMUM LOAD OF 900 PSF APPLIED.

equipment, including stoves, open fireplace grates, service water heaters, hand-fired domestic furnaces, hand-fired industrial furnaces, travelling grate stokers and in a hand-fired locomotive boiler. In general the combustion performance has been satisfactory to excellent. In a few instances where excessively high ash content material was used, some difficulty has been experienced in producing sufficient heat to meet the demand but this was not because of poor burning properties of the pelletized fuel. Detailed descriptions of a number of such tests are included in the report of Mulcey and Eckerd.[2]

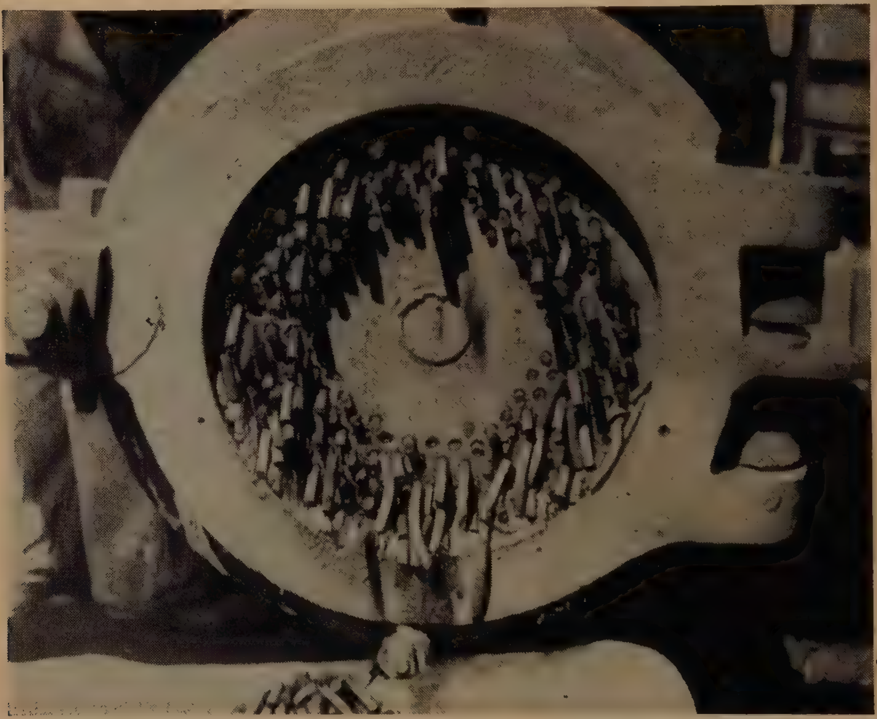

FIG 12—PILOT PLANT EXTRUSION PRESS PRODUCING ¾-IN. PELLETS FROM BITUMINOUS COAL SILT.

Various estimates by interested parties have been made as to the cost of upgrading the silt by extrusion. The process is still in the development stage and firm figures on equipment and operating costs have not yet been established. Neglecting the value of the silt, the extremes in current cost estimates range from $1.25 to $2.00 per ton of dried product. A 10 ton per hr plant complete with mixing, handling, extrusion and drying equipment has been estimated to cost between $40,000 and $65,000. The rate of amortization of this investment influences markedly the cost per ton of the product. Similarly the cost of the product will vary as much as 25 to 50 cents per ton depending upon whether or not a weatherproofing agent is required. Until such time as plant scale operations are conducted for a reasonable period of time, reliable cost estimates cannot be made.

The market value of the extruded product is similarly open to some question. Because of the relatively high ash content the product will probably not command a price equivalent to prepared sizes of the same fuel. The uniformity of size, high "apparent reactivity," low bed resistance and desirable burning characteristics make the product a useful fuel, however, and it is believed that a substantial market could be developed in both domestic and industrial applications. When silts with a naturally low ash content or silts cleaned to low ash contents are pelletized, it appears possible that the product may be able to command a price comparable to the prepared sizes of similar fuels.

CONCLUSIONS

Coal silts ranging in rank from subbituminous to anthracite have been successfully extruded without binders to produce pelletized fuel. Over two dozen different silts have been pelletized in the laboratory extrusion press, and several hundred tons of anthracite and bituminous coal silts have been pelletized in pilot plant operations.

The mechanism by which the silt particles are bound into stable pellets appears to be similar to that by which low rank coals have been successfully briquetted with pressure alone. This mechanism is believed to be one of capillary forces.

The two most important silt properties influencing the strength of the pellets appear to be size consist and moisture content. The size consist of the extruded product must be such that the capillaries produced upon extrusion are small enough so that capillary forces become effective. The moisture content must be such that every particle is thoroughly wetted and that the mass is plastic enough to flow through the die.

The extruded pellets normally contain from 13 to 20 pct moisture and in this condition are relatively weak. They may be safely handled, and piled after drying to about 10 or 11 pct. Further drying increases the strength almost as a direct function of decrease in moisture content. Satisfactory pellets, dried below about 6 pct moisture content, will withstand fairly severe handling without serious size degradation. The resistance to crushing and abrasion is about that of the so-called soft structure low and medium volatile bituminous coals.

All of the binderless pellets will disintegrate upon immersion in water and cannot be considered weatherproof. Anthracite and bituminous silt pellets may be waterproofed, however, by addition of small percentages of tars or waxes to the silt, and bituminous pellets can also be waterproofed by a mild low temperature carbonization.

The resistance to gas flow of a bed of pellets is lower than that of solid coal of comparable size and the "apparent reactivity" during combustion is greater than for anthracite of comparable size. During combustion the pellets will withstand imposed loads of at least 900 psf without significant disintegration or collapse. Fly material from burning pellets is slight.

The pellets can be produced in diameters up to at least 1 in., and normally range in length from about ½ to 1 in. for the ⅛-in. diam pellets up to 3 or 4 in. for the 1-in. diam size.

Power consumption for the extrusion press varies from about 12 to 20 kw-hr per ton of extruded product and is largely dependent upon the moisture content and size consist of the silt and the size of the extruded pellets.

Estimates of the cost of pelletization are hardly justified at the present time because application of the extrusion process to coal silts is too new. Based upon preliminary pilot plant data estimates ranging from about $1.25 to $2.00 per ton of dried product have been made. The market value of the product is similarly open to question although coal silt pellets have been used successfully in a variety of domestic and commercial burning equipment.

ACKNOWLEDGMENTS

The nature of the work has been such that the cooperation of numerous organizations and individuals has been necessary. The authors especially wish to express their thanks and indebtedness to R. C. Amero of the Floridin Co., Warren, Pa. for the use of the laboratory extrusion press, to C. J. Spangler of J. F. Pritchard and Co. for furnishing several of the pellet samples tested and for information on the pilot plant extrusion operations, to the engineers of the Anthracite Institute for their cooperation in furnishing samples, and, to K. M. Barclay, J. D. Clendenin, J. W. Eckerd, H. T. Grendon and Carl Nuebling, Jr. for material assistance in performing much of the experimental work. In part, the work reported was conducted as one phase of a cooperative research program, jointly sponsored by the Anthracite Institute and the Commonwealth of Pennsylvania, to discover new or improved uses for anthracite.

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DISCUSSION

(*H. F. Hebley and E. E. Finn presiding*)

C. V. SPANGLER*—DRYING AND GENERAL INTEGRATION OF PROCESS. The drying of the relatively soft extruded product, immediately as received from the extrusion die, is a necessary element of the process since the product cannot be handled in commercial quantities with any satisfaction at all until it has been hardened by evaporative drying. The drying equipment required will be a major part of any commercial installation. For a 10 ton per hr plant unit, heat drying equipment of perhaps two million Btu per hr input capacity will be required. The engineering details of the application of this heat to the product will vary with the product and location conditions.

In any event, the heat drying equipment will be a major unit of the plant installation. The labor requirement for attention and control will probably be equal to that required for the extrusion process proper. The fuel cost will be reasonable. As to installation cost, it is probable that the dryer will cost as much or more than the extrusion machine proper. The installation and operating cost figures contained in Dr. Wright's paper include the drier and drying costs, as well as the costs of other process elements and utility services.

It is to be pointed out that the extrusion process, to be successful in commercial operation, must operate on a continuous tonnage at a very regular and even rate. The essential mechanical elements are:

1. A stock bin and supply equipment.
2. A feeding device which can be readily adjusted, for any desired rate of feed of the raw fine coal.
3. Suitable and controllable equipment for the addition of water (or other additives) as required.
4. Mixing equipment to thoroughly distribute the additives through the moving charge going to the extruder.
5. The extrusion machine proper.
6. The drier, which will receive the product from the extruder.

Experience has indicated the absolute necessity of regular even feeding of the charge to the extruder, with a constant optimum moisture content in order to handle satisfactory tonnage, and to deliver a satisfactory product. The data presented by Dr. Wright clearly indicate the need for such control. The drier must take the extruded product, harden it by drying at the same tonnage rate, so that the resultant final product can be handled by normal means.

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* J. F. Pritchard and Co., Pittsburgh, Pa.

Full commercial scale pilot plant equipment is now in the process of being assembled and fabricated, under a cooperative arrangement, which will demonstrate and confirm all of the above outlined needs for the commercial use of the process. In the development of this process, it will be the function of J. F. Pritchard and Co. to design and build such equipment as will be needed, and to make such equipment, and engineering service as required, available to the coal industry. As far as is known at this time, the cost information submitted by Dr. Wright is substantially correct.

POSSIBLE GENERAL APPLICATIONS

In review it can be stated that considerable laboratory and pilot plant scale work have been done, and obviously much remains to be done. The small pilot plant has been used on both anthracite and bituminous coal with satisfactory results, taking into account the obvious mechanical limitations of the particular machine which was adapted for coal service. A brief review of possible general applications seems to be in order:

Anthracite Silts

There is a very considerable tonnage of silts accumulated, and still accumulating, in the anthracite region. A certain amount of these silts can be sold by the industry to large industrial and utility consumers who can afford to install necessary special combustion equipment to burn them. This market however cannot absorb anything like the total tonnage produced. It would seem practical and economical for anthracite producers to extrude their silts, and burn the product in their own boiler plants, since they have in many instances adequate market for the coal that they are now burning in considerable tonnage.

Bituminous Coals, Medium and Low Volatile

Several of these coals have been satisfactorily extruded. There is believed to be economic room for the extrusion of the fines from the central Pennsylvania fields of this class, for introduction into the Pittsburgh domestic market, which is now under a smokeless fuel ordinance. Some of the coals of suitable volatility are so friable that they cannot be satisfactorily delivered to this market without further processing.

Bituminous Coals, High Volatile

These coals, if not too strongly coking and swelling, can be partly carbonized at about 750°F as described by Dr. Wright. It is possible by this method to make a strong, waterproof, and smokeless product. Most coals of this class extrude well, and if waterproofed could be added to slack products to upgrade size.

Sub-Bituminous Coals and Lignites

The limited work done on this class, available in considerable quantity in the midwest, indicates that they extrude very satisfactorily. They do not however satisfactorily take up pitch, tar or asphaltic materials for waterproofing. Since they are mainly noncoking, they do not waterproof by the application of heat, as other bituminous coals react to carbonization. It is possible however to waterproof them by the addition of 5 pct by weight of Florigel (fullers earth) or other clay materials. These clay materials are added to the raw coal, mixed and extruded with it. The product is then heated to about 1000°F at which temperature the clay material "sets." The resultant product is water resistant. Obviously 5 pct ash has been added to the product; however if the moisture content of the original raw coal is high enough, as many midwestern coals are, the net result may be an upgrading of the coal.

Gasification and Blending of Coals

As Dr. Wright has pointed out, the "reactivity" of extruded coal is frequently higher than the "reactivity" (per cent of ash-moisture free coal consumed per unit of time) of the raw coal from which the product is made. This might very well be of advantage in any coal gasification process. It might also be practical to introduce catalytic materials into the mixture before extrusion. Further, since certain gasification processes cannot satisfactorily use a coking or strongly swelling coal, it may be practical to blend coals to get an optimum material for gasifying. The extrusion process offers a great many opportunities for the blending of coals for many specified uses.

J. W. ECKERD*—I have several comments that I would like to make. First, let me say, the paper is excellent and for a number of us who have been, for the past several years, investigating the process it has been very enlightening.

I am very much interested in the theory of pellet cohesion, that is, the capillary theory. We have found that a number of silt characteristics, particularly ash content, appear to influence this quality of the pellets. Whether these characteristics exert a direct influence or they are a result of, or a measure of, some other characteristic is difficult to determine. Nevertheless, we have never been able to make pellets with as high a resistance to impact and abrasion from low ash coals as we have from high ash coals.

We have never ground any coals to 100 pct −200 mesh and, hence, may never have reached the size consist required for maximum packing or minimum voids. We did grind one low ash coal to −65 mesh, but it did not appreciably improve the pellet quality. If the first pass of a high ash silt through the machine does not produce satisfactory pellets it is usually possible to improve their quality until they are satisfactory by recycling the material through the machine. This procedure has not been successful in producing satisfactory pellets from low ash coals.

It appears, from the information in this paper and results we have obtained with various silts, that the differences in pellet quality may be due to the wettability of the coal surfaces. If high ash coals are more readily wet than low ash coals, it would explain why the high ash coals produce better pellets than those with a low ash content and at the same time are more readily extruded.

There are several other points on which I wish to comment. We have extruded several bituminous coals in our laboratory. One of these coals presented no difficulty and produced excellent pellets. A second coal investigated was never successfully pelletized. Wetting agents, asphalt binder, and multiple passes were all used but no satisfactory pellets were produced. I believe Dr. Wright has had much the same experience with a number of bituminous coals.

The moisture content for each size of pellets and for each silt is critical. It has been our experience however, that better pellets can be produced at the relatively low moisture content of from 12 to 14 pct than at the higher moisture contents of from 18 to 20 pct often used. The pellets can be produced at approximately the same rate with the lower moisture content, but at almost certainly a higher power consumption.

The last point on which I wish to comment is mixing. We have used wetting agents and reduced pressure, both with variable success, as aids in mixing to attempt to improve the product and to facilitate extrusion. The mechanical difficulties encountered with the reduced pressure system made its evaluation difficult, but in one particular instance it did appear to have a beneficial effect. A number of wetting agents have been quite promising in that extrusion was more readily accomplished with certain silts and a better product was obtained. We have also found the pug mill to be a very useful tool in this important operation.

R. E. ZIMMERMAN*—I found Dr. Wright's paper very interesting. It so happens that the Hanna Coal Co. has done some experimental work on pelletizing their slurry at some of their cleaning plants, and I find that we concur with many of the things he has found out.

I cannot quite concur, however, with the idea that strength of pellets is largely effected by size consist of feed—within limits of course. We found, similar to Mr. Eckerd's findings, that the ash content, or more particularly the clay content of the coal has quite a lot to do with the strength of pellets produced. We have taken clean fines and made relatively poor pellets and taken high ash slurry and made good pellets. The clean fines, although somewhat coarser than the high ash slurry was not markedly coarser and a fact not generally realized, coals of similar friability tend to be pulverized in the extruder to approximately the same size consist after one pass, and certainly after a second pass through the machine.

The term high ash should be further explained to mean not inherent ash but free impurities such as colloidal clay, the latter obviously having cementing properties. It seems logical, also, that this is only one factor in the variation in the strength of pellets. The wettability of the coal surfaces may also be a governing element and freshly ground coal may have different strength than weathered or oxidized coal in storage.

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* Anthracite Institute.

* Hanna Coal Company, St. Clairsville, Ohio.

Another point which I believe should be mentioned on pelletizing coal fines is that unless the pellets are waterproofed they will disintegrate readily when they come in contact with water. Bituminous pellets may be carbonized satisfactorily. We find, however, that certain asphalt binders added at the rate of 1 to 2 pct did a very good job of waterproofing.

In general, I believe that the method described by the authors for pelletizing fine silts has considerable merit and possibilities. It would certainly seem worthwhile to continue investigating this method of upgrading fine sizes.

W. W. Everett*—Messers Day and Wright have covered the field as much as has been done up until now. This paper is a complete account of what is known to date of a method of making pellets out of coal silt by an extrusion process. The authors have ably presented the scope of the problem and the details of the process, and have pointed out the possible use of the finished product both as a domestic and as an industrial fuel.

The work done so far demonstrates that it is possible to upgrade coal silt by an extrusion process, and make pellets which are particularly suitable for use as fuel in the hand-fired furnace.

In regard to the behavior of this product in a furnace, it is important to note two distinct advantages that the authors claim this fuel has over regularly sized coal. First, it offers less resistance to the gas flow through the fuel bed.

Second, it has superior combustion behavior as regards thermal decrepitation and apparent reactivity, and is more free burning than a sized product. These two factors are very important in combustion and gasification applications as the authors point out in the paper.

You will note from the contents of the paper that the process of making pellets out of coal silt is quite complicated and involved. The extrusion alone requires a high degree of skill.

The moisture content and size consist of the feed to the extrusion machine are critical, and are also very difficult to control. When the pellets are extruded from the machine they are delicate and fragile and must be handled with extreme care until dry.

Equipment for handling and drying these green pellets is still in the development stage. While there are many difficult engineering problems involved, the American manufacturers' ingenuity should certainly be able to solve them.

The authors' estimate of the potential supply of silt available for making pellets, namely, 10 pct of the total anthracite production as of 1943 can be considered a little high because there has been a general increase in the tonnage of No. 4 and No. 5 recovered from the breaker silt during the intervening years.

In the Northern Anthracite Field it is our estimate, based on spot tests, that the quantity of raw silt available at the present time is about 5 pct of the total production. This silt ranges in size from ⅓₂ to zero, and runs from 30 to 35 pct in ash content.

It seems quite evident from a producer's standpoint that any silt which is to be made up into pellets must first be processed to reduce the ash content to as near 12 pct as possible in order that the product may have any customer appeal.

Such a process will recover about two-thirds of the raw silt for the production of pellets. The waste product will still have to be disposed of by means of back slushing inside the mines or in silt dams located on the surface.

If for any reason there should occur a persistent drop in the present market demand for No. 4 and No. 5 buckwheat, these sizes would be available for pelletizing; and this would very materially increase the quantity and develop an active interest in this process of upgrading coal silt by means of extrusion into pellets. This process certainly appears to have more promise of success than any other process so far developed.

E. R. Kaiser*—I would like to hear from either Dr. Wright or one of the others who have been working on this with regard to what the economic potentials are.

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* Glen Alden Coal Company, Wilkes-Barre, Pa.

* Bituminous Coal Research, Pittsburgh, Pa.

How much is the pellet worth in relation to other forms of anthracite? What are the economics of pelletization? Can you give us a general picture of costs that we may get a better idea of the potentials of this method?

C. C. Wright (authors' reply)—Mr. Eckerd can answer the question better than I because he has been contacted just recently by some groups interested in marketing the pellet if he can make it, so I will ask Mr. Eckerd to answer the question.

J. W. Eckerd—Unfortunately I do not feel that I am in a position, at present, to make public the price that was discussed.

E. E. Finn—May I say, having been in the operating end for years, that one of the problems is the necessity of eliminating that fine material from streams. In Pennsylvania there is a law prohibiting stream pollution. Second is the cost of your normal disposal of these fines which you can write off.

Then there is the question of marketing, first in using the pelletized material possibly at the mines to replace sizes that are marketable and then what price you can get for this material for different uses.

C. C. Wright—I think I can go a little beyond what Mr. Eckerd had to say; the value of the pellets will be somewhere in the vicinity of sized fuel of the same type. Pelletized fuel seems to have very desirable combustible properties and even though it has a higher ash content these desirable properties are such that it should command a price comparable to the size fuel with which it is competing.

J. W. Eckerd—I think that perhaps I may go a little further than I did. The price discussion was in connection with the use of pellets as a specialty fuel. It was considerably above the price of domestic prepared sizes of anthracite at the present time. I do not know how large the particular market under discussion would be.

P. F. White*—May I add that during the development of the use of the pellets in several colliery power plants in the region, the pellets were handled up to eight times from the time they left the die until fired onto the fuel bed.

During these growing pains, the request was passed back that the pellets be made tougher in order to withstand the abuse. This was accomplished, so that toward the end of that phase of the project, the pellets held their shape without benefit of binder. Of course they were more porous and they burned more freely than standard colliery fuel, which is rice or barley or smaller.

One of the aims of the project was the possible development of a streamline operation in which pellets would be made and burned right at the colliery plants and thus release standard colliery fuel for domestic consumption.

Finally, the pellets were fired on traveling grate stokers where they did an excellent job. An equivalent number of pounds of pellets was burned, as compared with coal, per thousand pounds of steam developed.

E. R. Kaiser—I did several years of work on coal briquetting without binder and I might suggest something which perhaps these people have tried.

Rather than using the extrusion press, with material of high water content, a reciprocating plunger might be used to force coal of lower moisture through a die. The die could offer enough resistance to make a suitable briquetting pressure possible if it is not too wet. If the coal is fine and has some moisture in it, it might not be necessary to use vacuum deairing.

The reason I suggest the possible work with less water, using reciprocating elements rather than the extrusion press, is that there would be less of a drying problem; the ash material or clay in the coal would still act as a binder and the amount of water would still be sufficient and would help hold the pieces together.

It might be possible to do the whole thing with a lot less water and cut down on the drying expense.

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* Anthracite Institute.

Estimates of Moisture Increases Due to Water-spraying Coal for Dust Control

By T. W. Guy,* Member AIME

(Cincinnati Meeting, October 1947)

The increased moisture due to water-spraying for coal dust control is of interest even for mines from which the coal is to be wet-washed before screening, because in many cases wet coal dust materially decreases the loading efficiency of men and equipment at the faces. It also causes a serious increase in the normal wear and deterioration of all equipment with which it comes in contact. It is important to the consumer if it increases his as-received moisture.

No attempt will be made in this paper to cover the entire subject of water-spraying of coal for dust control. It will be confined to discussion and data on (1) percentages of dust in bituminous coals, (2) moisture increases due to spraying for dust control in mines or plants by methods which have proved satisfactory for periods of one to five or more years, and (3) a few suggestions for research.

Size and Percentage of Coal Dust

The U. S. Bureau of Mines[1] classifies as *coal dust* all bituminous coal passing a 20-mesh sieve (0.0328 in. square opening). The percentage of such dust to be found in ROM coal, of course, varies widely with the characteristics of the seam, and with the methods used in the mining and in the preparation of the coal.

ROM screen analyses of 12 U. S. coals selected at random from personal files given in Table 1 show:

20-mesh × 0 sizes as pct ROM average .... 6.3 pct Range 2.3 to 9.5 pct
20-mesh × 0 sizes as pct ⅜ × 0 sizes average.................................. 18.9 pct Range 12.7 to 23.9 pct

A few other screen analyses examined indicating 40 pct or more of 20-mesh × 0 dust in ⅜ in. × 0 sizes were not included because of uncertainty that they were representative. The number of results in Table 1 is not sufficient to give a reliable estimate of the average, or of the range to be expected, but it indicates that in every case the coal dust is a relatively large percentage of the *total ROM coal*, and, of course, a progressively larger percentage of resultants as the larger sizes are separated for marketing.

Table 1—*Percentages of Dust* (20 *M.* × 0 *Sizes*) *in ROM and Resultants*
(Data from TWG Files)

State	Pct. ROM	Pct. ¾ in. × 0	Pct. ⅜ in. × 0
Pa..........	3.9	10.5	16.7
Pa..........	8.2	14.8	22.2
Pa..........	8.7	13.3	17.8
Pa..........	9.5	13.0	18.6
W. Va......	3.3	14.3	23.0
W. Va......	8.0	12.2	19.8
W. Va......	8.0	13.7	19.8
W. Va......	8.1	11.6	13.3
Ill..........	2.3	8.1	12.7
Ill..........	4.4	14.2	18.9
Ill..........	4.6	14.5	20.2
Ill..........	6.8	19.8	23.9
Average.....	6.3	13.3	18.9
Low.........	2.3	8.1	12.7
High........	9.5	19.8	23 9

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* Chairman AIME Coal Preparation Committee. Consulting Engineer, Charleston, W. Va.

[1] References are at the end of the paper.

Percentage of Dust in Cuttings

Little or no data were obtained on size consist of machine cuttings. An article by Cree and Connor[2] in 1938 showed for a series of 13 cutting tests in one seam size consist in which the smallest sieve used was ¼ in. These showed percentages of cuttings averaging as follows:

Inches	Per Cent	Range Per Cent
Plus 1	14.0	5.0 to 19.8
¾ × 1	6.9	2.3 to 22.3
½ × ¾	22.1	6.2 to 33.4
¼ × ½	22.1	18.0 to 25.3
¼ × 0	35.9	21.9 to 46.0
20 M × 0 sizes estimated from the test data as pct of cuttings average	14.0	7.0 to 16.0

The cuttings as pct of total coal vary with $\frac{\text{thickness of kerf}}{\text{total thickness of coal}}$ for horizontal kerfs, and $\frac{\text{thickness of kerf}}{\text{width of place}}$ for vertical kerfs. For a 6-in. horizontal kerf in a 30-in. seam, the percentage would be 20 pct and in a 90-in. seam, 6.7 pct. If the pct of cuttings in a given ROM is 10.0 pct and the dust in the cuttings as pct of cuttings 24.0 pct or, as pct of ROM 2.4 pct and, if the total dust in this coal as pct of ROM is 6.0 pct then the dust in the cuttings as pct of total dust in the ROM would be 2.4 pct/6.0 pct or 40.0 pct.

This would give 60 pct of the dust in this ROM as originating from sources other than the cuttings. These figures are not given as generally applicable, but to illustrate a point on which more data are needed.

Moisture Increase Due to Water Sprays for Dust Control

A study of a number of Bureau of Mines publications, data published by The American Mining Congress, and others in the U. S. and abroad, shows underground spraying for dust control in a considerable number of mines ranging from less than 1 gal to more than 7½ gal per ton of ROM coal, giving an *increase in moisture* as pct of ROM from *less than* 0.4 *pct* to *more than* 3 *pct.*

They also show spraying of cuttings ranging from less than 1 gal to 60 gal or more per ton of cuttings, corresponding to moisture increase as pct of cuttings from *less than* 0.4 *pct* to *more than* 20.0 *pct.*

Four examples within these limits are as follows:

Example A

In the Jan. 1942 number of *Mechanization,* J. A. Saxe[3] of Island Creek Coal Co. describes apparatus and methods used for dust control underground with the use of water and a wetting agent.

An average of 1.5 gal of spray liquid is used per ton of ROM coal on cutters, loaders, cars, dumps and transfer points. This corresponds to a calculated increased moisture in the ROM of 0.63 pct.

With this spraying the dust reduction is from about 300 million to 30 million particles per cu ft, and is frequently around 20 million particles at the machine operator's station. It is understood that with this spraying it is not generally necessary to use further sprays for dust control in the screening and preparation plants.

Eight Island Creek preparation plants make screen separations at ¼ in. or smaller, 6 use dedusters for removing −48-mesh dust, and 6 use pneumatic processes for cleaning −¼-in. coal. The increased moisture due to sprays for dust control underground in these mines does not seriously increase the difficulties of screening and pneumatic cleaning the ¼-in. × 0 coal. It is understood that the same system has been continued in these mines since that time, and is operating satisfactorily at present.

Example B

After several years of experimental work at this mine, all cutters, universal track type, were equipped with integral water tanks and spray pumps, and very satisfactory results were obtained in both undercutting and shearing with a jet applied to the ingoing side of the cutter chain. Plain water used averaged 1⅔ gal per ton of cuttings, equivalent to an increased moisture as pct of the cuttings, 0.7 pct = 1.0 pct of ⅜-in. × 0.

Dust sampling showed less than 15 million particles per cu ft at the cutting machine operator's station. The 0.7 pct increased moisture in the cuttings did not seriously interfere with screening at ⅜ in. and smaller, or with the pneumatic cleaning and dedusting at this plant.

In order to eliminate increase in water due to wear of the discharge orifices, experiments are being made with a low capacity, high pressure turbo pump which will supply a constant quantity of water to the spray nozzles.

Example C

In a large group of mines, some without mechanical cleaning plants, some using wet washing including the fines which are later heat dried, some wet washing without the fines, some air cleaning and dedusting the fines, a great many methods and variations have been tried in water spraying for dust control underground, on the surface and in the preparation plants.

It has been found from this group that any surface moisture in excess of 3 pct in the coal shipped is enough to cause serious trouble in cold weather, and is at any time likely to cause complaints from the consumer.

It has been found desirable for efficient pneumatic cleaning, dedusting, and so on, to use water sprays on dry coal in order to decrease the variability of the feed, although a substantial part of the feed may be too wet for efficient cleaning. The electric eye and other devices have been used to turn on sprays when the coal is dusty, and to cut them off when the dust is within permissible limits.

In most mines of this group, sprays are used on the cutter bars and in some cases on loaders. They are also used on dumps, cars, transfer points, crushers, and the like, where the conditions require it.

These mines on an average use from ½ to ¾ gal of water per ton of ROM coal. This corresponds to the moisture increase as pct of ROM of about 0.2 to 0.3 pct. For ⅜ in. × 0 this is equivalent to about 0.6 to 1.0 pct.

Tests show that in low moisture coals the ⅜ in. × 0 Rd. sizes containing surface moisture averaging 3 pct, the surface moisture in the 14-mesh × 0 sizes in the same sample will be about 5 to 8 pct, and in the 48-mesh × 0 from the same sample probably 10 to 12 pct. At these moistures the fines are still not wet and will not stick to dry surfaces with which they come in contact. When the average surface moisture in the ⅜ in. × 0 reaches 4½ to 5 pct, the moisture in the fines becomes high enough so that they begin to wet dry surfaces with which they come in contact and begin to stick to such surfaces.

100± samples of heat dried and of air cleaned coal showed surface moistures in the −14-*mesh sizes* as follows:

	Heat Dried, Pct	Air Cleaned, Pct
Average	2.2	7.2
Minimum	1.1	4.8
Maximum	4.1	11.5

For total moistures add 1.5 pct to the above.

Example D

This mine has not used sprays underground except at conveyor transfers in dry sections.

Plain water sprayed per ton, underground, as pct ROM estimated less than

0.10 pct. In preparation plant, water with wetting agent applied by nine misting sprays with automatic controls at rotary dump, weigh basket, and ROM conveyor discharge, measured quantity average per ton, 0.2 gal. Moisture increase per ton ROM 0.08 pct. Total moisture increase due to underground and plant sprays as pct ROM 0.18 pct. Dust reduction by sprays estimated 70 to 80 pct.

This system has been operating satisfactorily in this plant for 6 or 7 yr. In this plant all sprays are applied to ROM coal. If it is assumed that ⅔ of the water sprayed goes into the −⅜-in. sizes, the moisture increase in the ⅜ in. × 0 would be 0.58 pct.

SUMMARY

20-mesh × 0 coal classified by Bureau of Mines as dust in different ROM coals possibly ranges as pct ROM from about 2 to ±15 pct as pct of the ⅜ in. × 0 sizes ±10 to ±40 pct.

Insufficient data were available to the Committee for estimate of pct of dust in cuttings, and in dust from each of other principal sources. A guess is that dust from cuttings as pct of total dust will range ±15 to ±40 pct.

Dust control examples *A* to *D*, inclusive, show that with care and skill in spraying, with water and a wetting agent, or, with plain water, the dust has been controlled in a considerable number of mines through periods 1 to 6 yr, or longer, reducing dust at critical points from 70 to 90 pct, and meeting requirements then in effect, with *increased moisture* in *ROM from* 0.08 *to* 0.8 *pct* and in *⅜ in.* × 0 *from* 0.6 *to* 1.5 *pct.*

Some mines report that surface moistures above 3 pct in products shipped will cause serious trouble in freezing weather, and are at any time likely to cause complaints from customers because of low Btu values. One reports trouble as low as 3 pct total moisture.

Surface moistures above 4½ to 5 pct in ⅜ in. × 0 sizes will generally cause serious trouble, (1) in screening, dedusting, pneumatic cleaning, and so forth in the preparation plant, and (2) in consumers' handling facilities.

Critical Moisture

Surface moistures increasing with decreasing size above ⅜ in. can be handled successfully up to what might be called the *critical moisture.* This "*critical moisture*" may be defined as the surface moisture at which the coal *begins* to wet dry surfaces with which it comes in contact. The little data available indicate that the critical surface moisture in −48-mesh and smaller sizes may be as high as 15 pct or even more.

Important reductions in the surface moisture variability of slack coals can be made by putting them through a blending or mixing bin. Even a comparatively small surge bin materially improves the moisture uniformity of the feed to pneumatic equipment and to fine coal screens.

Additional Data Needed

It is believed that additional information on the following points will be helpful to the industry in planning dust control to achieve maximum dust prevention without making the coal sticky:

1. Sources of dust: Weight percentage and size consist of dusts made by drilling, cutting, breaking, loading, miscellaneous handling and transfers, and by crushing.
2. More data on the wettability of different coals, which is known to vary widely.
3. Data as to use and efficiency of wetting agents.
4. Data as to the critical surface moisture in the ⅜-in. × 0 and smaller sizes, and particularly in the dust sizes, 20-mesh and smaller, in coals from different mines and of different classifications. 20-mesh × 0 coal with zero surface moisture may be able

to take 10 ±pct surface moisture before reaching the critical or sticky stage.

With skill in the planning and operation of a water spray system, with wetting agent where needed, it is believed that coal dust can be controlled within reasonable limits, and at the same time the serious difficulties of handling wet coal, can be avoided in many cases.

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The Thermal Drying Of Fine Coal

By Orville R. Lyons,* Junior Member, and A. C. Richardson,* Member AIME

(New York Meeting, February 1948)

During the past few years there has been a growing demand by coal operators for detailed information about the performance characteristics of the various dryers now being manufactured, preferably in such a form that the information could be used to determine the type of dryer best suited to a particular drying problem. A few operators have been desirous of going even further, because they are of the opinion that present-day dryers are not properly designed and that fundamental studies of dryer design should be undertaken.

This growing demand for a method of dryer evaluation and the indicated need for fundamental research resulted in the sponsorship by Bituminous Coal Research, Inc. of a project at Battelle Memorial Institute to investigate the thermal drying of coal.

In investigating this problem, the most feasible method of attack appeared to be a survey of active operations to determine the present status of the thermal drying of coal. Data were obtained, usually by actual plant tests with the cooperation of the various preparation engineers and their staffs, for 9 screen-type dryers, 8 rotary-type dryers, 3 continuous-pallet-type dryers, 2 flash-type dryers, 1 tray-type dryer, and 1 cascade-type dryer. The testing program provided: (1) data for 24 coal-drying operations, (2) information about the inherent characteristics of the coals dried, (3) information about the previous treatment that the coal had received, and (4) information about operating costs.

The data obtained through the dryer survey and relationships determined from the data make it evident that all of the types of dryers included in the survey can be operated with the same degree of effectiveness if each type of dryer is used to dry the sizes of coal for which it is best suited. It should be noted that it is usually undesirable and often impossible to dry different sizes of coal in one type of dryer. By use of the data in the report, any prospective dryer user, having determined the surface moisture percentage and screen-size analysis of the coal to be dried, the tons of water to be removed per hour, and the final surface moisture percentage desired, should be able to determine with a fair degree of accuracy: (1) the difficulty of the drying problem, (2) the approximate cost of drying, (3) the required dryer inlet gas temperature and volume and exhaust gas temperature and volume, (4) the type of dryer to use, and (5) the size of dryer required.

The method of testing the dryers and determining the difficulty of drying can be used by operators already having dryers to check the performance of their particular dryer and determine whether they are operating effectively.

Considerably more information was obtained from the survey of thermal drying than was anticipated when the survey was first considered. However, a great deal of work still remains to be done in the field

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and in the laboratory before a complete answer can be provided to all questions concerning the thermal drying of coal.

The Survey Procedure

Attempts to make even a simple evaluation of dryer performance on the basis of data published in the literature failed because the data were incomplete. It seemed evident that an evaluation of dryer performance, and especially attempts to compare one type of dryer with another, would require very complete and reliable data. It was decided, therefore, to obtain data for every portion of the drying plant as accurately as possible, even if the data appeared valueless at the time. The "Suggested Recommended Practice for Testing Drying Equipment," issued by the ASME in 1933, was used as a guide and additions were made, such as screen-sizing data, that appeared especially desirable in coal-drying analyses.

The dryer operators supplied general information about each particular dryer and the coal being treated. Moisture contents of the wet and dried coals were determined by taking 30- to 50-lb samples at half-hour intervals during a day's run, determining total moisture for each sample, and calculating an average value. The same samples were then used for screen-analysis purposes.

Capacity of the dryer was determined whenever possible by timing the loading of railroad cars and obtaining weights for the timed cars. In some cases it was necessary to stop belts and conveyors periodically and weigh the material contained on a measured number of feet of belt or certain number of flights. Then, by determining belt speed, tonnages could be calculated. In other cases chutes were inserted and the dryer product was weighed. In every case at least three weights were obtained in each series.

Furnace and dryer temperatures were determined by means of a portable, direct-reading potentiometer, which was recalibrated at weekly intervals to insure accuracy. Gas volumes were determined by two methods, Pitot tube and CO_2 determinations. The CO_2 determinations plus knowledge of the total carbon content of the coal burned in the furnace and the number of pounds of coal burned per hour allowed calculation of gas volumes with greater accuracy than the Pitot tube method.

At the majority of the plants the coal burned per hour in the furnace was determined in cubic feet per hour by measuring the feed-bin cross section and the pull-down per unit of time, and then converted to pounds by determining the weight of a cubic foot of the furnace feed. The furnace feed was sampled for proximate analysis, total carbon, and Btu determinations at each of the plants tested.

Power requirements were determined by having the plant electrician determine the amperage and voltage requirements of each piece of equipment in the drying plant while in operation.

Operating cost was the hardest item to obtain since it had to be taken from the operator's records, and in many cases the method of keeping the records was such that drying costs could not be determined.

Evaluation of the Field Survey Data

Completion of the field tests on operating dryers and tabulation of the data obtained achieved the immediate objectives of the survey, but mere possession of the data did not accomplish the ultimate objectives of a better understanding of dryer operation, a method for comparing one drying operation with another, and improvements in dryer operation and design. While it was evident that the field survey provided a fund of data on coal drying never before available, it was also apparent that the data needed careful study in order to determine whether they could be used to accomplish the ultimate objectives.

The Factors Affecting Dryer Performance and Drying Costs

Table 1 contains the basic data obtained for the 24 dryers included in the survey and also contains the most complete data available in the literature for installations drying either coal or other materials.[1,2,3,4] After careful consideration of the data contained in Table 1, it was decided that certain items appeared to be related to dryer performance, others appeared to be related to drying costs, and others had no connection with either dryer performance or drying costs.

The factors considered as affecting drying rate or dryer performance were: (1) the surface moisture percentage of the wet coal, (2) the surface moisture percentage of the dried coal, (3) the weight of inlet gases, (4) the temperature of the inlet gases, (5) the temperature of the exhaust gases, and (6) the average particle size of the coal being dried. Surface moisture was considered to be critical, rather than total moisture, because at the majority of drying plants it was considered unnecessary to dry the coal to a moisture content lower than the seam or surface moisture content of the coal as mined.

The factors considered as affecting the cost of drying were: (1) power requirements per unit of water removed, (2) heat input per unit of water removed, (3) total amount of water removed or the capacity of the drying installation, (4) man-hours of labor required per unit of water removed, and (5) maintenance requirements. Items 4 and 5 were not included in Table 1, because labor requirements per unit of water removed were nearly equivalent for the majority of the dryers tested and maintenance requirements could not be determined by means of a short testing period.

To check the truth of the assumptions that certain factors affect dryer performance and other factors affect the cost of drying, it was necessary to correlate each group of factors with a valid reference value. In the past, the reference value usually used for comparing dryers on the basis of effectiveness has been the thermal efficiency of the operation, where thermal efficiency is the ratio, expressed in per cent, of the heat required to evaporate a unit weight of water to the heat input per unit weight of water for the given dryer. However, attempts to analyze the dryers by means of thermal balances indicated that calculated thermal efficiencies were in error by as much as 25 pct and that, therefore, thermal efficiency was not a good reference value. Further, to complicate the problem, the amount of water removed by evaporation could only be approximated for the majority of the screen-type dryers tested because considerable water was also removed by drainage. Finally it was determined that the use of the total Btu input to the furnace per unit of water removed, rather than the water evaporated, in the dryer provided a correlative value for the majority of the factors thought to influence dryer effectiveness.

The heat input per ton of water removed depends upon the accuracy of the methods used to determine the amount of coal burned, the heat value of the coal burned, the tonnage of coal being dried, and the moisture determinations on the wet and dried coals. While extreme accuracy cannot be claimed for all of these items, it was believed that the methods of sampling and analysis were sufficiently accurate to allow the use of the heat input per ton of water removed as a reference value.

Apparently the inherent design or method of operation of screen- and continuous-pallet-type dryers is such that any advantage gained by removal of water by drainage is lost by increased radiation and unaccounted-for losses.

The actual operating cost of drying in dollars per unit of water removed was considered to be a sufficient reference value

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[1] References are at the end of the paper.

TABLE 1—*Dryer Operating Data for Plants 1–24 from Survey, A–J from Taggart,*[1] *X and Y from Rosin and Rammler,*[2] *Z from Colliery Guardian*[3] *and from Mott*[4]

Plant No	Type of Dryer	Seam Moisture, Wt Pct	Surface Moisture In Feed, Wt Pct	Surface Moisture In Product, Wt Pct	Water Removed, Tons Per Hour Per Plant	Btu Input In Fuel Fired Per Ton Of Water Removed	Kwhr Consumption Per Ton of Water Removed		Inlet Gases, Lb per Ton of Water Removed	Inlet Gases, Cfm At Existing Temperature And Pressure, Per Plant	Exhaust Gases Cfm At Existing Temperature and Pressure, Per Plant	Temperatures, Degrees F		Average Particle Size Of Feed Material, In.	Dried Product, Tons Per Hr	Operating Cost of Drying, Dollars Per Ton of Water Removed				
							Fan and Dryer Only	Total				Inlet Gases	Exhaust Gases			Fuel	Power	Labor	Maintenance	Total
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)	(14)	(15)	(16)	(17)	(18)	(19)	(20)	(21)
1	Rotary	6.5	5.0	−0.8	4.35	4,200,000	29.0	34.7	16,416	42,000	42,000	933	194	0.2883	66	0.54	1.72	0.36	0.13	2.75
2	Screen	16.4	4.0	−0.5	8.24	3,190,000	15.2	19.2	16,968	74,000	66,000	795	104	0.4774	145	0.45	0.36	0.62	0.45	1.88
3	Flash	9.5	18.0	−2.7	2.58	4,980,000	13.5	17.0				1147	257	0.0642	9					
4	Rotary	1.9	10.9	1.2	4.46	5,000,000	8.0	25.4	20,988	52,000	39,000	873	167	0.2886	40	0.56	0.20	0.29	0.38	1.43
5	Rotary	1.9	9.0	0.5	3.50	5,350,000	10.1	32.4	19,884	39,000	31,000	872	192	0.3210	37	0.65	0.24	0.36	0.44	1.69
6	Screen	1.9	9.9	1.3	6.43	3,400,000	15.0	17.8	17,262	56,000	58,000	746	145	0.3540	68	0.46	0.17	0.27	0.26	1.16
7	Screen	1.5	6.0	1.6	5.64	2,390,000	7.3	10.6			30,000	600	150	0.6472	117					
8	Rotary	12.0	9.2	1.1	4.38	3,120,000	9.9	13.8	12,942	35,000	28,000	1014	150	0.2401	42					
9	Rotary	14.5	8.4	0.5	7.04	3,500,000	6.1	8.6	10,398	51,000	31,000	1227	166	0.1821	68					
10	Revolving Tray	16.0	10.3	1.1	13.80	2,740,000	3.3	5.4	16,044	127,000	55,000	917	134	0.2194	112					
11	Rotary	2.25	12.5	5.0	12.75	2,740,000	9.4	15.5			80,000	1467	155	0.1353	147	0.28	0.14	0.31	0.23	0.96
12	Screen	9.5	7.0	2.0	11.22	2,700,000	15.9	19.7	27,000	171,000	147,000	900	97	0.5664	188					
13	Screen	1.2	5.0	2.3	1.22	7,680,000	32.4	36.5	69,840	46,000	49,000	836	91	0.4414	42					
14	Screen	1.1	10.4	3.8	4.01	1,790,000	10.1	11.2	10,476	22,000	32,000	786	93	0.3377	55					
15	Rotary	6.75	9.6	1.5	14.49	3,310,000	6.4	7.7	8,118	85,000	75,000	1275	145	0.2178	150	0.47	0.14	0.25	0.69	1.55
16	Screen	10.0	10.1	3.1	12.40	1,610,000	12.6	14.5	24,192	141,000	147,000	662	85	0.3046	141					
17	Screen	16.0	5.8	2.3	8.58	2,390,000	21.8	28.6	10,980	54,000	50,000	897	118	0.4914	192					
18	Rotary	13.0	3.2	0.6	2.82	4,510,000	22.0	26.0	18,510	31,000	20,000	958	155	0.3712	90	1.30	0.26	0.78	0.83	3.17
19	Cont. Pallet	9.0	12.1	8.7	2.38	2,540,000	12.0	15.4	13,362	17,000	21,000	805	97	0.2412	56	0.49	0.23	0.98	0.65	2.35
20	Cont. Pallet	9.0	8.3	3.2	4.02	3,670,000	11.7	15.6	15,000	30,000	28,000	736	103	0.2487	65	0.34	0.16	0.67	0.45	1.62
21	Cont. Pallet	10.0	6.8	1.0	5.27	3,930,000	8.9	11.9	12,240	32,000	30,000	732	106	0.2565	76	0.30	0.14	0.60	0.40	1.44
22	Cascade	9.0	9.9	2.2	2.82	2,800,000	5.3	7.4	14,256	20,000	16,000	732	124	0.2713	30					
23	Screen	14.0	3.0	1.3	2.26	9,540,000	40.5	43.6	67,170	64,000	78,000	556	61	0.5638	109					
24	Flash	1.9	9.4	2.9	2.30	5,000,000	51.9		43,824	51,000	27,000	756	161	0.0502	29					
A	Rotary				0.55	3,200,000	17.6								10.0	1.07	0.23	0.55	0.03	1.88
B	Rotary				6.40	5,680,000	7.2								80.0	1.89	0.10	0.08	0.03	2.10
C	Rotary				1.20	4,240,000	13.7								15.0	1.42	0.18	0.18	0.03	1.81
D	Rotary				2.90	3,120,000	18.0								50.0	1.04	0.24	0.10	0.01	1.39
E	Rotary				0.80	3,000,000	23.2								15.0	1.00	0.31	0.20	0.04	1.55
F	Rotary				2.00	3,420,000	7.0								5.5	1.01	0.09	0.08	0.03	1.21
G	Rotary				0.57	4,760,000	23.5								1.5	1.59	0.31	0.35	0.16	2.41
H	Rotary				8.00	3,620,000	7.9								55.0	1.21	0.10	0.04	0.03	1.38
J	Rotary				2.05	4,640,000	23.1								7.5	1.55	0.31	0.10	0.05	2.01
X	Flash				4.40	4,090,000	33.0					842	158	(−⅛ inch)	38.5					
Y	Flash				7.80	3,600,000	23.2					1112	158	(−1/32 inch)	35.0					
Z	Cascade				2.50	4,288,000	5.9					788	188	0.1620	39.5					

for the factors thought to influence drying costs.

In Fig 1–6 the data contained in Table 1 for the various factors thought to influence dryer effectiveness are plotted against heat input per ton of water removed for the corresponding dryer. Fig 1 also represents the data presented in Table 2 which has been abstracted with some modifications from a Table in Kent's Mechanical Engineering Handbook, 11th Ed., sec. 3, p. 51, N. Y., 1936, John Wiley and Sons.

A considerable degree of correlation exists between the majority of the various factors and the heat input per ton of water removed. The data obtained from the survey agree fairly well with the data obtained from Kent's Handbook. Fig 1, 2, 3, 5, and 6 reveal expected results, while Fig 4 reveals that inlet gas temperatures alone have very little influence on dryer effectiveness.

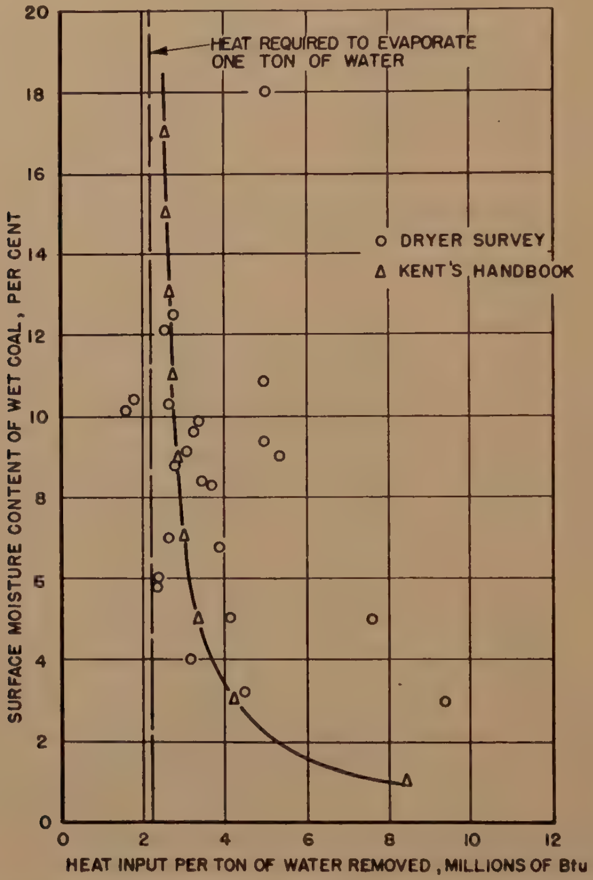

Fig 1—Relation of surface-moisture content of wet coal to heat requirements.

Only a rough correlation exists between the factors thought to influence drying costs when plotted against their respective drying costs. Both power consumption and heat input show considerable variation from straight-line relationship with their respective costs. Fig 7 reveals that the operating cost of drying per ton of water removed varies with the amount of water removed, decreasing as the tons of water removed increase, until a critical point is reached in the range of 6 to 8 tons of water removed per hr. Removing more water or increasing the size of the unit, once the critical point has been reached, apparently will no longer result in a decrease in cost.

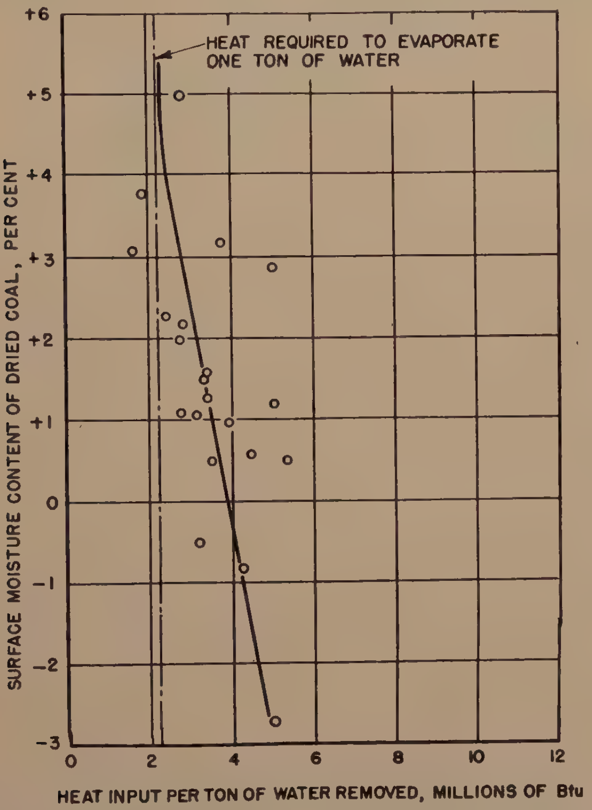

FIG 2—RELATION OF SURFACE-MOISTURE CONTENT OF DRIED COAL TO HEAT REQUIREMENTS.

TABLE 2—*Water Evaporated and Heat Required for Drying**

Moisture in Material to be Dried, Pct	Lb Water Evaporated per Ton of Dry Material	Btu Required for Drying	
		Per Ton of Coal Dried	Per Ton of Water Evaporated
1	20.2	85,624	8,477,600
3	61.9	130,424	4,214,000
5	105.3	180,936	3,436,600
7	150.5	231,560	3,077,200
9	197.8	284,536	2,877,000
11	247.2	339,864	2,749,700
13	298.9	397,768	2,661,500
15	352.9	458,248	2,597,040
17	409.6	521,752	2,547,600

* Kent's Mech. Eng. Handbook, 11th Ed., Sec. 3, p. 51, N. Y., 1936, John Wiley and Sons.

The correlations obtained between the assumed effectiveness and cost factors and their respective reference values, even though not exact, are of value because they substantiate the correctness of the assumptions and also provide information concerning the exact effect of each of the various factors. The correlations, however, do not provide a ready method for comparing one dryer with another.

Comparing Dryers on the Basis of Effectiveness

Considerable time was devoted, without much success, to attempts to compare dryers by means of rating scales based on the five factors affecting dryer performance. Then it was decided to approach the problem from the point of view of a coal operator having a drying problem but not having a dryer. This hypothetical coal operator would know or could readily determine certain facts about his drying problem. Relatively simple tests could be used to determine (1) seam and total moisture percentages and, by difference, the surface moisture content of the proposed wet feed, (2) the tonnage of wet feed to be dried per hour, and (3) the screen analysis of the coal to be dried. Then the operator could arbitrarily set the final surface moisture content desired, based on market requirements, and could calculate the total tons of moisture to be removed per hour. This information is all that would be readily available and it then must be used to determine the type and size of dryer needed and the cost of drying.

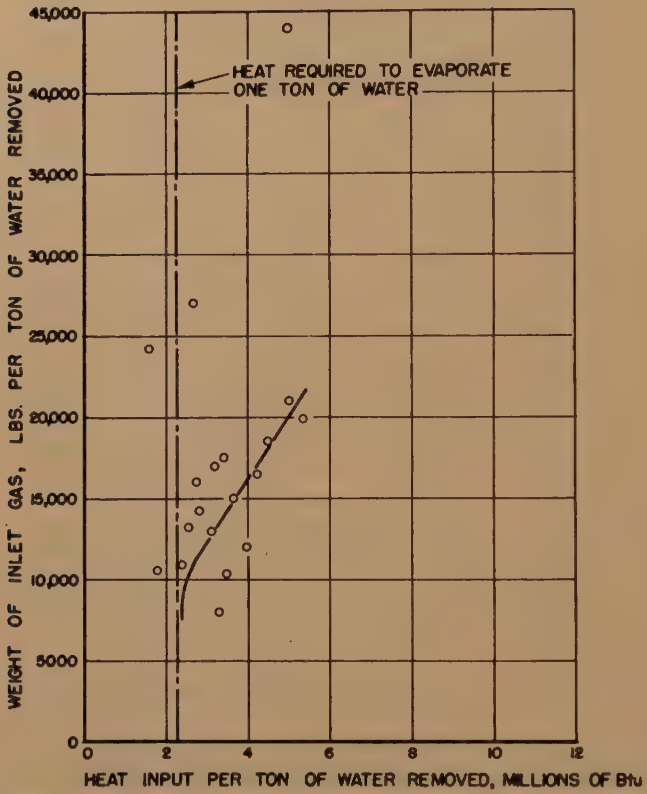

FIG 3—RELATION OF INLET GAS WEIGHT TO HEAT REQUIREMENTS.

By this method, three of the factors previously found to influence dryer effectiveness are immediately available. These are the surface moisture content of the wet coal, the surface moisture content of the dried coal, and the average particle size of the coal to be dried. Assuming that they are the only factors necessary in determining dryer effectiveness, since the other factors can only be determined by means of tests on operating dryers, it should be possible to correlate them as a unit with the heat input required per ton of water removed. Because the three factors represent two different dimensions, it is not possible to add them together, but it is possible to multiply them and thus to obtain a numerical factor which should be proportional to the difficulty of the drying problem, if the primary assumption is correct.

The final surface moisture percentages for a number of the dryers tested were negative values; that is, the coal was dried to a point below its seam moisture content, and hence it was necessary to change these negative values to positive ones, without changing their relative position with respect to the positive final surface moisture values, prior to using them in any multiplication procedure. One method that appeared usable was that of adding to the final surface moisture figures any value sufficiently large to provide positive sums and then dividing these sums by the same value. This method would provide positive numbers without changing the relationships already existing. It was decided to use 40 as the value since its use would result in a positive number, even if lignites were being included in the calculations. For example, with Dryer 1 the sum of 40 is added to −0.8 to provide 39.2

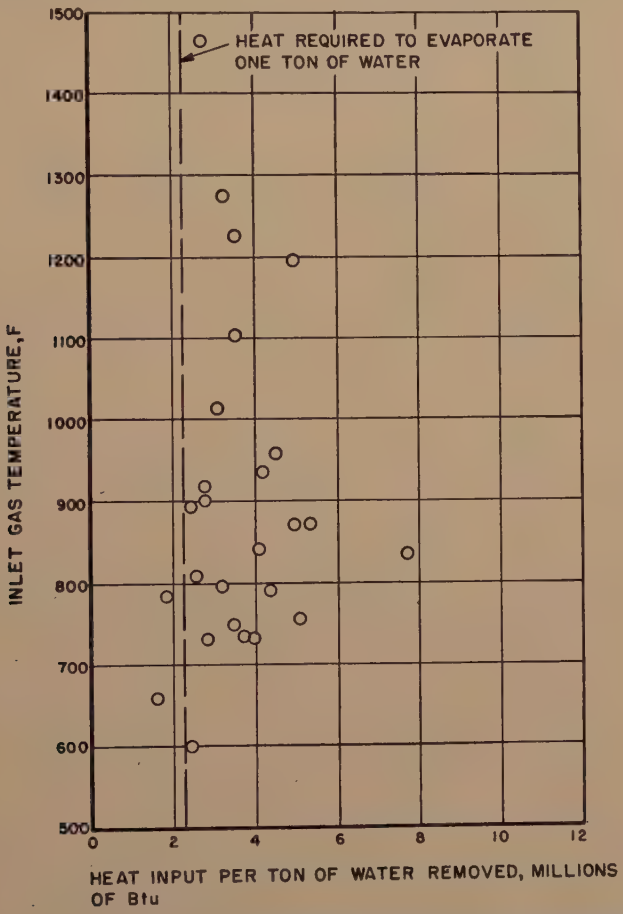

FIG 4—RELATION OF INLET GAS TEMPERATURE TO HEAT REQUIREMENTS.

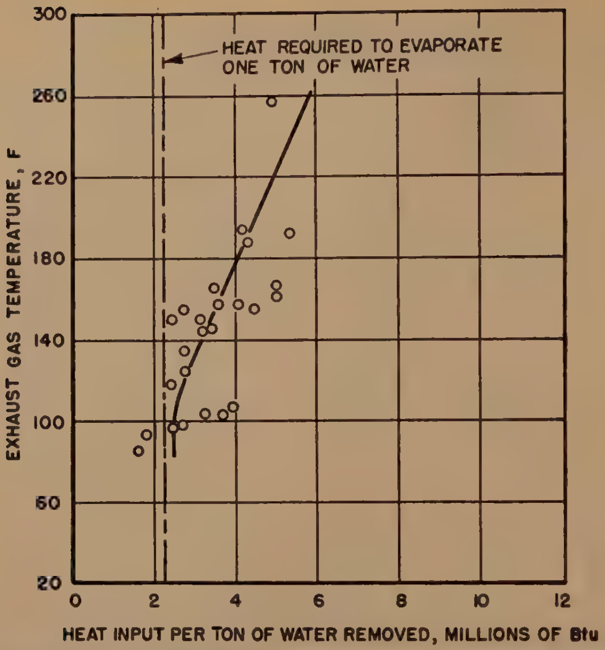

FIG 5—RELATION OF EXHAUST GAS TEMPERATURE TO HEAT REQUIREMENTS.

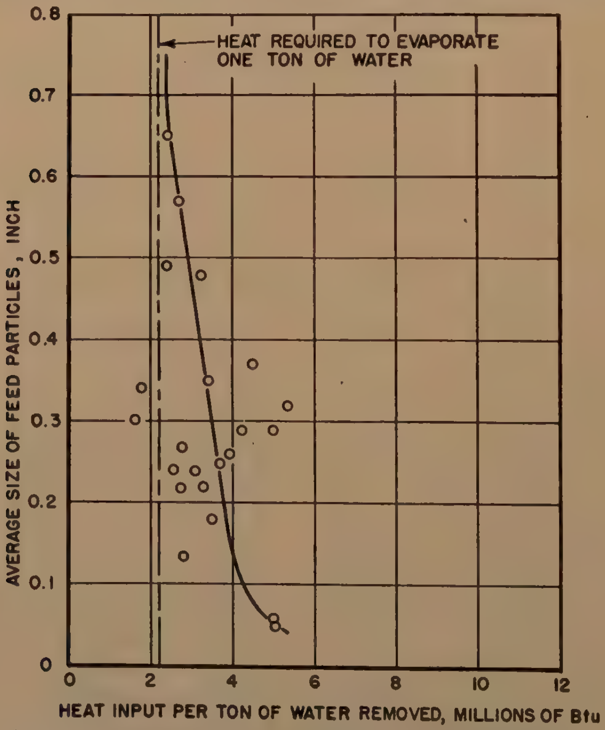

FIG 6—RELATION OF PARTICLE SIZE TO HEAT REQUIREMENTS.

which in turn is divided by 40, resulting in a final product of 0.98.

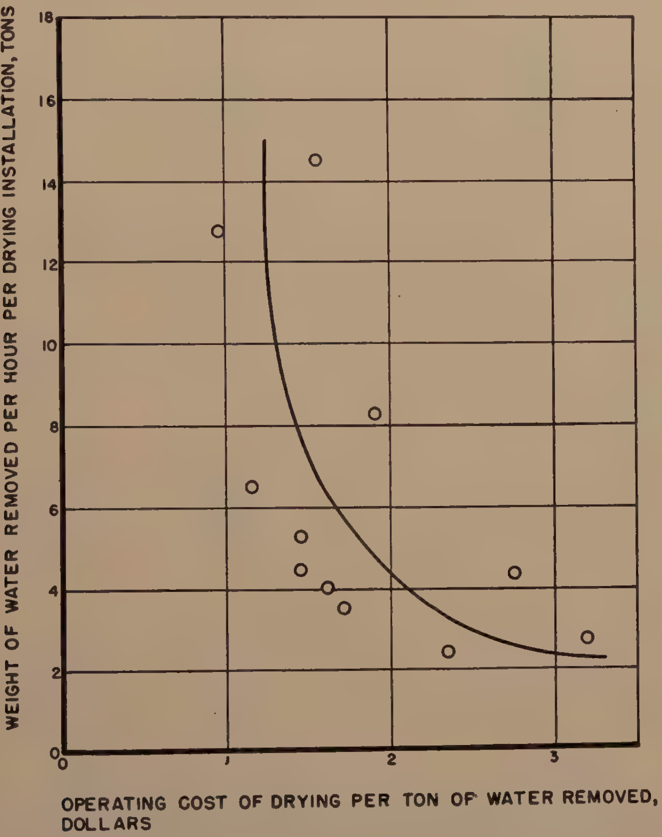

FIG 7—RELATION BETWEEN AMOUNT OF WATER REMOVED AND COST OF DRYING.

Table 3 contains data for the dried coal surface moisture factor, based on 40, and also contains the product of the three available effectiveness factors. This product has been called the numerical ease of drying. The following examples are provided to clarify the method used in determining the numerical ease of drying:

Example 1: A difficult drying problem would be one in which a fine coal is being treated to remove all of the surface moisture present in the coal. For example, to dry a wet coal having an initial surface moisture of 10 pct and an average size of 0.065 in. to a final surface moisture of zero pct would be considered difficult.

The numerical ease of drying is obtained by multiplying the surface moisture content of the wet coal, or 10 pct by the surface moisture factor for the dried coal, or $\frac{40 + 0}{40}$ which is equal to 1, by the average size of the coal, or 0.065 in.

Then the numerical ease of drying

$$= (10)\left(\frac{40 + 0}{40}\right)(0.065) = 0.65$$

Example 2: A relatively easy drying problem would be one in which a coarse coal is being treated to remove only a part of the surface moisture. For example, to dry a wet coal having a surface moisture content of 8 pct and an average size of 0.45 in. to a final surface moisture of 4 pct would not be considered difficult.

TABLE 3—*Dryer Evaluation Data*

Plant No. (1)	Type of Dryer (2)	Dried Coal Surface Moisture Factor, Surface Moisture Pct Plus 40 Divided By 40 (3)	Numerical Ease of Drying (4)	Inlet Gas Weight, Lb Per Plant (5)	Btu Input to Dryer Per Min. (W) $(t_1 - t_2)$ (0.24) (6)	Screen Area, Sq Ft (7)	Cylinder Area, Sq Ft (8)	(Numerical Ease of Drying) × (Tons of Water Removed Per Hr Per Installation) (9)
1	Rotary	0.98	1.41	1190	211,000		1130	6.1
2	Screen	0.99	1.89	2330	390,000	301		15.6
3	Flash	0.94	1.08				472	2.8
4	Rotary	1.03	3.24	1560	264,000		965	14.5
5	Rotary	1.01	2.91	1160	190,000		965	10.4
6	Screen	1.03	3.62	1850	267,000	182		23.3
7	Screen	1.04	4.04			120		22.8
8	Rotary	1.03	2.28	945	196,000		1760	10.0
9	Rotary	1.01	1.54	1220	311,000		1760	10.8
10	Revolving tray	1.02	2.31	3690	695,000			31.9
11	Rotary	1.10	1.86				5210	23.7
12	Screen	1.05	4.16	5050	972,000	377		46.7
13	Screen	1.06	2.34	1420	255,000	126		2.9
14	Screen	1.09	3.83	700	116,000	126		15.4
15	Rotary	1.04	2.17	1960	492,000		4190	31.4
16	Screen	1.08	3.32	5000	695,000	341		41.3
17	Screen	1.06	3.02	1570	293,000	346		25.9
18	Rotary	1.01	1.20	870	168,000		1865	3.4
19	Cont. pallet	1.22	3.56	530	90,000			8.5
20	Cont. pallet	1.08	2.23	1005	153,000			9.0
21	Cont. pallet	1.02	1.78	1075	162,000			9.4
22	Cascade	1.05	2.82	670	98,000			8.0
23	Screen	1.03	1.74	2530	300,000	240		3.9
24	Flash	1.07	0.50	1680	242,000		236	1.2

The numerical ease of drying = (8) $\left(\frac{40+4}{40}\right)$ (0.45) = 3.96

By this method of calculation, the more difficult the drying problem, the smaller is the value for the numerical ease of drying.

Fig 8 shows the calculated numerical ease of drying for the various dryers plotted against the effectiveness reference value, heat input per ton of water removed. It seems apparent from a study of Fig 8 that there is a definite, good correlation between the numerical ease of drying and the heat input per unit of water removed.

Data for dryers 4, 5, 13, and 23 failed to agree with the data for the other dryers and a study of the test data and notes taken during the tests furnished the following explanations. Dryers 4 and 5, through no fault of design or construction, were operated with intermittent feed conditions and, even though the furnaces were operated so as just to maintain a fire during the no-feed periods, the periods were so numerous that the heat input per ton of water removed was higher than it would have been with a more continuous operation. Dryer 13 had its by-pass stack damper so badly rusted that, for practical purposes, the damper was nonexistent, and as a result, approximately one-half of the heat input to the system was wasted. Dryer 23 had exceedingly low inlet and exhaust gas temperatures and it is possible that the initial low evaporative effect plus recondensation of moisture might be in part responsible for the high heat input per unit of water removed. Another factor that deserves consideration is the low initial surface moisture content of the feed material. As shown by the data in Table 2, the heat input required per unit of moisture removed increases rapidly as the moisture content decreases. Since it was not practical to spend much time investigating the seemingly poor performance of dryer 23, the exact reason, or reasons, for the high heat input per unit of water removed were not determined.

A study of Fig 8 definitely indicates that dryers cannot be compared directly with one another unless the dryers being compared are handling identical feeds insofar as surface moisture content, size consist, and tonnages are concerned, and produce dried products having identical surface moisture contents. Evidently dryers must be compared with a theoretical or empirical 100 pct effectiveness based on the difficulty of the drying problem. In this instance, making some allowance for possible unavoidable errors in the data, dryers providing data that plot close to the curve shown in Fig 8 are considered to be operating efficiently, while dryers that do not plot close to the curve are operating inefficiently, and it appears evident that all of the types of dryers included in the

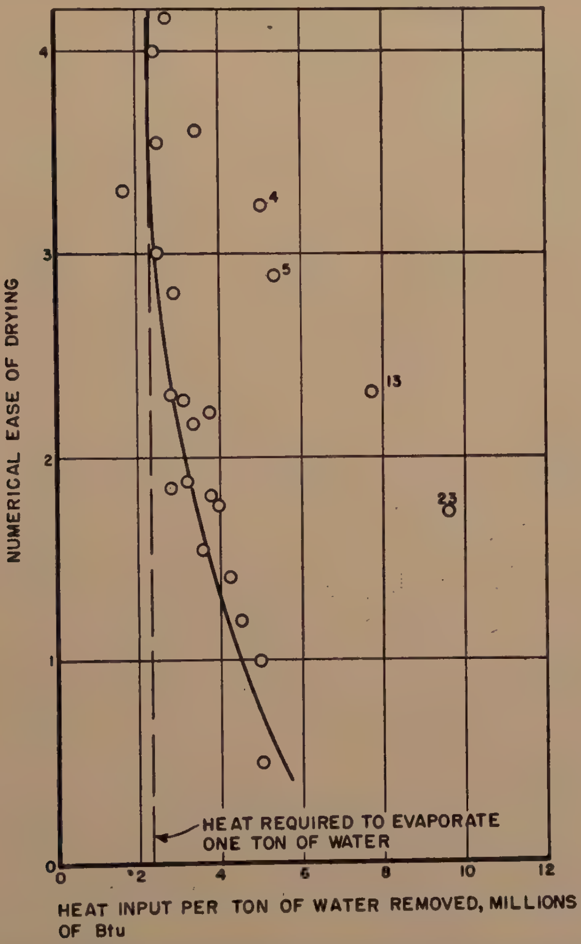

Fig 8—Relation between difficulty of drying and heat requirements.

survey can be operated with the same degree of effectiveness.

Predetermining the Cost of Drying

Having determined the difficulty of the drying problem, our hypothetical coal operator can estimate within fairly reasonable limits the heat input required per ton of water to be removed. Having already calculated the total tons of water to be removed per hour, he now possesses information about two of the factors affecting cost. The other factors, power consumption, labor, and maintenance requirements were assumed to be the same for all operations, on the original assumption that it must be possible to predetermine drying costs on the basis of the information available to a prospective dryer user. A relation is now required between these two established factors and the operating cost.

Fig 9 shows the relation between the product of the numerical ease of drying and the tons of water to be removed per hour to the operating cost of drying in dollars per ton of water removed. The data for these two values are taken from Tables 3 and 1, respectively. This graph is very similar to that contained in Fig 7, but shows a greater degree of correlation and therefore is more usable.

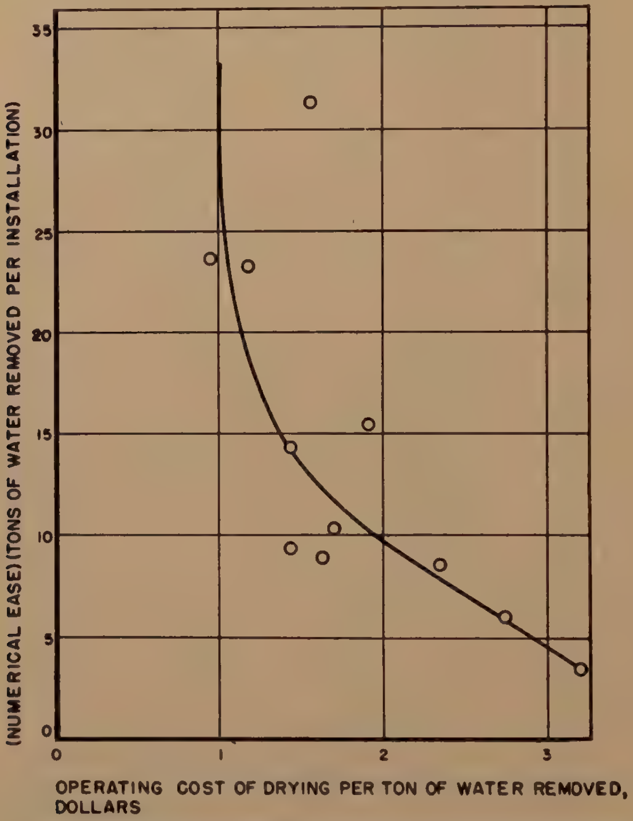

FIG 9—RELATION BETWEEN COST FACTORS AND COST OF DRYING.

Now our hypothetical coal operator can determine, with at least some degree of accuracy, the operating cost of drying his coal based on its drying difficulty and the tons of water to be removed.

Predetermining Dryer Conditions

While attempting to set up a standard for dryer effectiveness, several correlations were discovered which were of interest, but had no immediate application. One of these was an apparent straight-line correlation between the surface moisture content of the dried coal and the exhaust gas temperature as shown in Fig 10. Another one was a correlation between the tons of water removed per unit of time per dryer installation and the heat input to the dryer calculated by multiplying the inlet gas weight per unit of time by the difference between the inlet and exhaust gas temperatures and by the specific heat of the gases as shown in Fig 11. The first relationship is not well known, or at least has never been discussed in the literature, while the elements of the second one are commonly known. The data for these relationships are taken from Table 3.

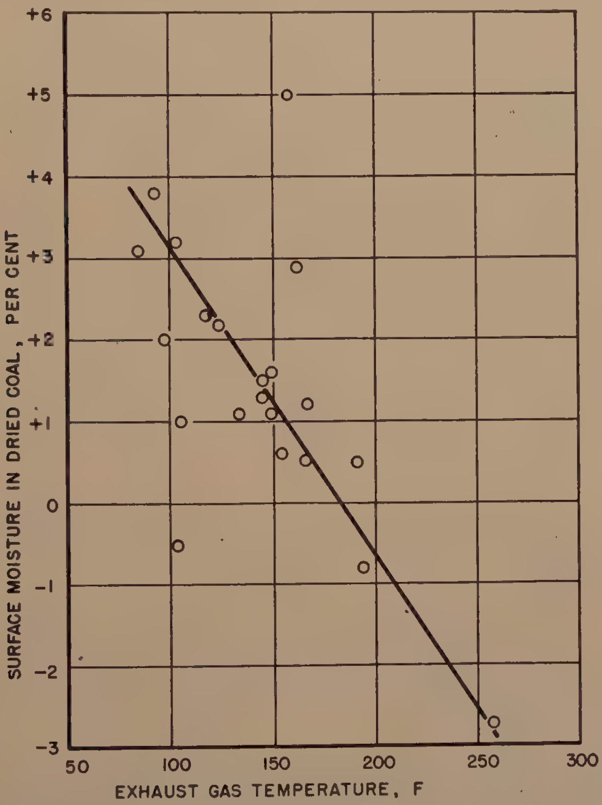

FIG 10—RELATION BETWEEN SURFACE-MOISTURE CONTENT OF DRIED COAL AND EXHAUST GAS TEMPERATURE.

Our hypothetical coal operator, in setting the final surface moisture desired in the dried coal, also evidently set the exhaust gas temperature that must be maintained. It should be noted that it may not be desirable to maintain so high an exhaust gas temperature as shown necessary to obtain the desired moisture reduction without making provision for a cooling section to prevent later ignition of the coal.

Then, knowing the tons of water to be removed and the exhaust gas temperature, and arbitrarily setting the inlet gas temperature around 900°F, since contact with the wet coal will immediately reduce the gas temperature, thus protecting exposed metal surfaces while the furnace can usually stand temperatures in the range of 800 to 1100°F without excessive maintenance costs, the weight of gas required can be calculated. By this method the exhaust gas temperature, inlet gas temperature, and inlet gas weight can be determined. The weights of the inlet and exhaust gases are the same in any air-tight system and should not differ greatly in a well constructed and well-maintained dryer. Hence, either the inlet gas volume or the exhaust gas volume can be calculated knowing the inlet gas weight and the inlet and exhaust gas temperatures. Fig 12 contains graphically the information necessary for converting gas weights to gas volumes at any given temperature. Dryer operation will be controlled by the exhaust gas temperature and the size and speed of the hot gas or exhaust gas fan.

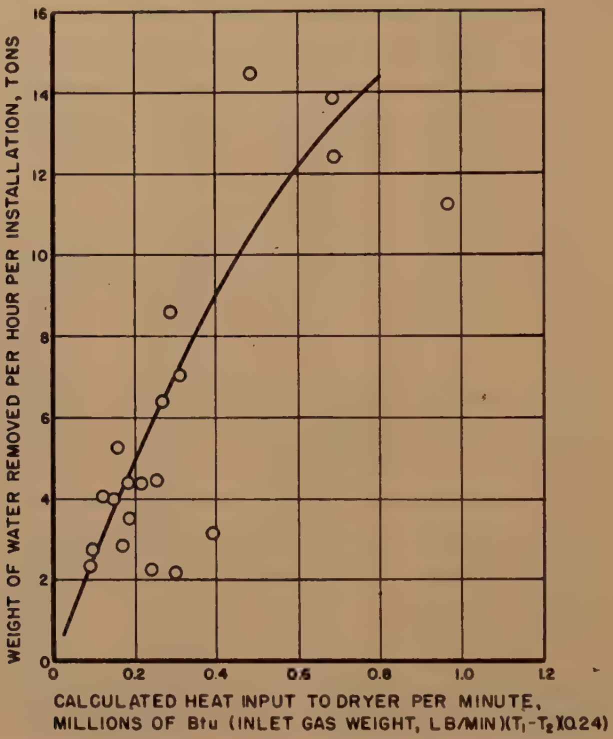

FIG 11—RELATION BETWEEN TONS OF WATER REMOVED AND HEAT INPUT TO DRYER.

Once the dryer and furnace setup have been designed to comply with any given set of dryer conditions it is difficult to increase the load on the system while it is relatively easy to operate at a lighter load. This fact should be given careful consideration because it is quite common in coal preparation practice for the load on existing equipment to be suddenly increased, sometimes as much as 100 pct or more. Slight increases can be taken care of by increasing furnace temperature and thereby the inlet gas temperature, or by speeding up the fan to increase the gas flow. Large increases in load on the dryer are bound to produce an unsatisfactory product. A lightened load is easily taken in stride because as the load drops off the exhaust gas temperature will tend to rise, and either automatic or manually operated controls can be utilized to reduce the stoker feed rate to the furnace and thus reduce the total heat input.

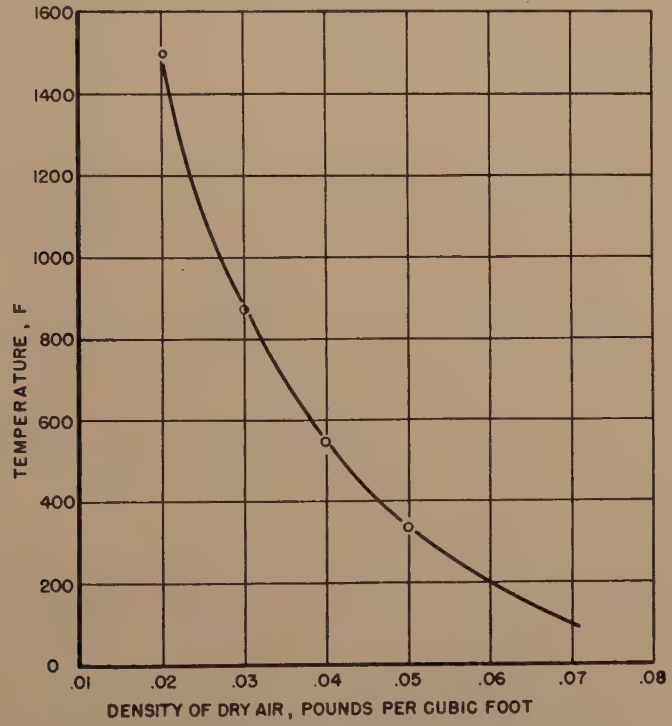

FIG 12—RELATION BETWEEN TEMPERATURE AND DENSITY OF DRY AIR.

Predetermining the Type and Size of Dryer

Our hypothetical coal operator, now quite well supplied with information about dryers, is still desirous of determining the type and size of dryer required for his particular drying problem. Fig 13 shows the average particle size of the feed material in inches plotted against the type of dryer used to dry that particular size. It is readily apparent that the three main types of dryers are being used on definite size ranges which show little tendency to overlap. The miscellaneous types of dryers have the same range as the rotary dryers and therefore could be used in their place. The choice between a rotary-type dryer and a cascade-, tray-, or continuous-pallet-type dryer will depend upon the initial cost of the equipment and the operating economics possible with the particular equipment.

The size of dryer required can be determined from Fig 14 and 15, for screen and rotary dryers only, by knowing the tons of water to be removed per hour and determining the required area from the proper graphs. In Fig 14 the area is in sq ft of screen and can be used directly, while in Fig 15 the area is cylinder area and must be converted to diameter and length dimensions by selecting any desired diameter between 7.5 and 12 ft, using 0.5-ft increments, and then calculating the necessary length. It should be noted that these figures represent average values and that the individual points show considerable variance from the average and, therefore, the relationships are not exact. However, they do provide a rough method for determining the size of dryer required. Insufficient data were obtained, owing either to a lack of installations or to inability to test existing installations, to provide similar correlations for the other types of dryers.

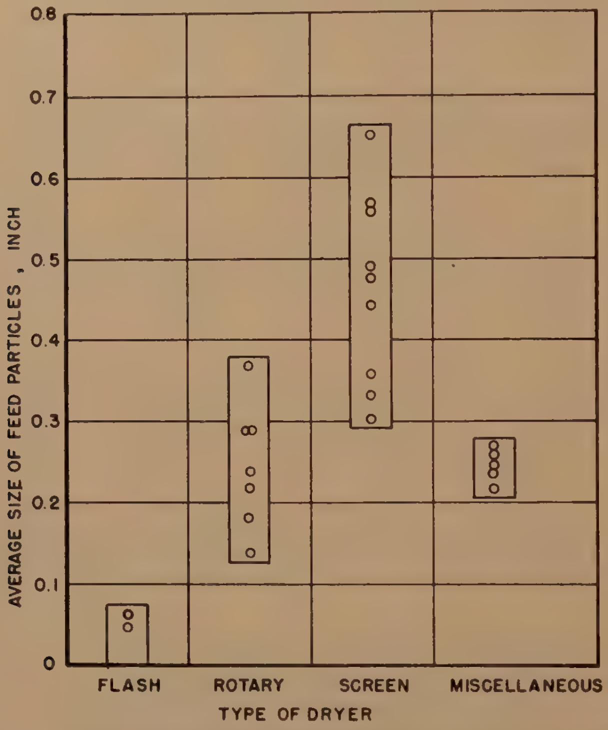

Fig 13—Relation between size of particle and type of dryer.

Conclusions

Any prospective dryer user, by means of a few simple tests on his particular coal, using the relationships determined from the dryer survey data, should be able to determine, with a fair degree of accuracy:

1. The difficulty of the drying problem.

2. The approximate cost of drying in dollars per ton of water removed. This is based on the assumption that the operation will be an efficient one. 3. The required exhaust gas temperature as determined by the final surface moisture content of the dried coal 4. The inlet gas temperature. 5. The inlet or exhaust gas weight and volume. 6. The type of dryer to use. 7. The size of dryer required if either screen or rotary dryers are to be used.

The method of testing the dryers and the numerical ease of drying can be used by operators already possessing dryers to check the performance of their particular dryers and determine whether they are operating effectively.

The survey data indicate that:

1. Thermal efficiency, based on the ratio of the heat required to evaporate a unit weight of water to the heat input per unit weight of water removed by evaporation in a given dryer, was found not to be a significant and useful value. Thermal efficiency does not take into account the numerous other important variables involved in any drying problem. Furthermore, accurate determination of the thermal efficiency is extremely difficult in field tests of many dryers.

2. The total heat input to the furnace per unit weight of water removed in the dryer was found to correlate with most of the factors thought to influence dryer effectiveness, and apparently is a usable indicator of dryer effectiveness.

3. All of the types of dryers included in the survey evidently can be operated with the same degree of effectiveness if each type of dryer is used to dry the sizes of coal for which it is best suited and the difficulty of the drying problem is given proper consideration.

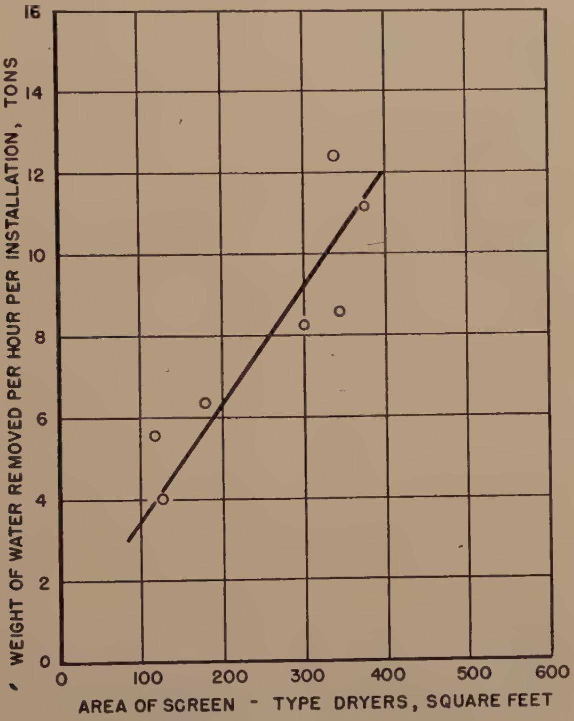

FIG 14—RELATION BETWEEN TONS OF WATER REMOVED AND SCREEN AREA.

4. Screen-type dryers are being used to dry the coarser sizes of coal treated. Inclusion of very fine coal in the feed results in blinding of the screen and an unsatisfactory operation.

5. Rotary-, tray-, cascade-, and continuous-pallet-type dryers are being used to dry the intermediate sizes of coal.

6. The flash dryer is the only dryer now being used to dry very fine coal.

7. The operating cost of drying ranges from $0.08 to $0.18 per ton of dried coal produced, with the majority of the dryers having costs of approximately $0.10 per ton.

8. The operating cost of drying ranges

from $1.00 to $3.50 per ton of water removed, with the majority of the dryers having costs of approximately $1.50 per ton of water removed.

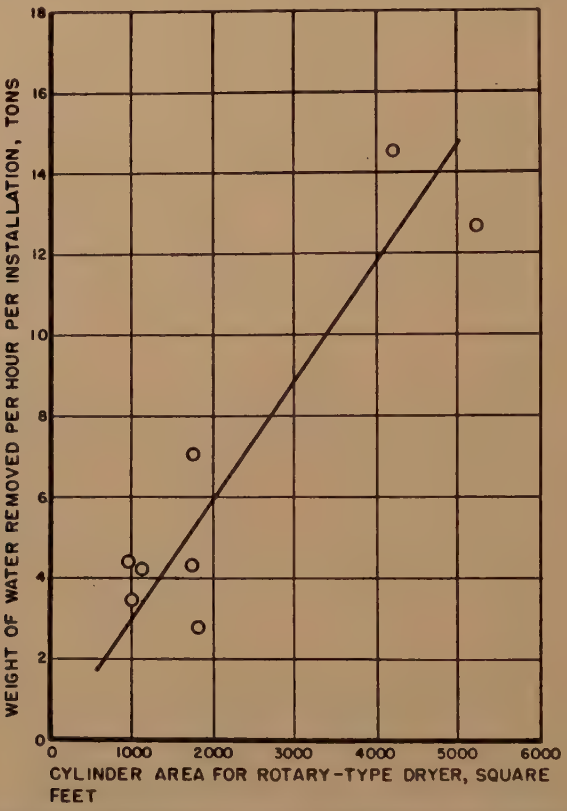

FIG 15—RELATION BETWEEN TONS OF WATER REMOVED AND CYLINDER AREA.

9. Capital charges, based on the cost of the dryer furnace, the dryer, the necessary ductwork, exhaust fan, and exhaust stack, for a plant having a one-shift operation for 200 days each year, and assuming that the plant will be completely depreciated in 10 years, range from $0.04 to $0.18 per ton of dried coal produced, or $0.44 to $1.85 per ton of water removed. The cost of conveyors, buildings, and auxiliary equipment such as centrifuges is not included in the above figures. These may, and often do, exceed the cost of thermal drying equipment and will increase the capital cost accordingly.

10. Maintenance is a major item representing from one-third to one-half of the total operating cost of drying. Maintenance costs in many cases could be reduced by: a. Coating exhaust ducts, fan housings, and exhaust stacks to prevent corrosion. b. Use of corrosion and fly-ash-resistant fan impellers. c. Use of stainless steel for exposed steel surfaces, such as screens in a screen-type dryer. d. Adequate controls to prevent overheating of furnaces and dryer inlet ducts. e. A well-planned maintenance program.

11. Dryer performance could be improved in many instances by: a. Eliminating heat losses due to air infiltration. b. Operating the dryer at a steady rate. c. Making periodic check tests to determine dryer effectiveness. d. Checking control

instruments at regular intervals. e. Having an intelligent dryer operator.

12. Future work should include laboratory, pilot-plant scale experiments to check the evaluation procedures developed from the dryer survey, to determine the exact effect of the factors influencing dryer effectiveness, and to determine the reasons for the empirical relationships developed from the dryer survey data.

ACKNOWLEDGMENTS

This work would not have been possible without the willing cooperation of the various coal companies and their preparation engineers. The authors wish to express their appreciation to J. W. McDivitt and James J. Merle of the Ayrshire Collieries Corp.; F. H. Bird of the Binkley Coal Co.; R. H. Sherwood and R. G. Baughman of the Central Indiana Coal Co.; James Hyslop and R. E. Zimmerman of the Hanna Coal Co.; R. G. Nunn and Lynn Caton of the Hickory Grove Coal Mining Corp.; E. C. Carris of the Island Creek Coal Co.; Hugh B. Lee, Jr., of the Maumee Collieries Co.; K. R. Bixby, Robert Ronk, and W. H. Weimer of the Midland Electric Coal Corp.; V. G. Leach, William Burnett, and Elroy J. Kraetsch of the Peabody Coal Co.; E. F. Stevens and Laning Dress of the Pyramid Coal Corp.; H. F. Hebley, J. F. Laubach, and S. M. Parmley of the Pittsburgh Coal Co.; F. P. Calhoun, Dr. C. J. Potter, and George Kennedy of the Rochester and Pittsburgh Coal Co.; Howard Herder, Paul Halbersleben, and J. A. Bottomley of the Sahara Coal Co.; Thomas C. Cheasley and Albert Massman of the Sinclair Coal Co.; T. Howard Green and W. C. Adams of the Tecumseh Coal Corp.; and R. L. Sutherland and Byron Somers of the Truax-Traer Coal Co.

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DISCUSSION

(*D. H. Davis and T. W. Guy presiding*)

E. R. KAISER*—The survey by Messrs. Lyons and Richardson was sponsored by Bituminous Coal Research, Inc., to give the coal industry and others a factual report of the latest commercial practice in the art of drying coal. The authors have accomplished the objective insofar as it was reasonably possible to do so and the final report will be published.

The report gives a prospective dryer user the information he would need in making an appraisal of the technical factors and economic possibilities of an installation at his mine. The costs given are general, but each particular prospective installation can be estimated from cost data furnished currently by the manufacturers of dryers and auxiliaries and from general contractors who would make the installation.

The authors do not reach a final conclusion on overall costs but give operating costs and capital charges, the latter not fully complete. However, combining the figures they have given, which may be low as the capital costs are not complete, the total cost of drying coal in the plants surveyed ranges between 12 and 30.5 cents a ton of dried product. The total cost of removing water ranges from $1.60 to $5.38 a ton of water.

Assuming an installation in which 16 tons of wet coal are dried to 15 tons of dried coal, a loss of 133⅓ lb per ton of dried coal, the purchaser of the coal would derive an economic benefit for which he should be willing to pay more for the dry coal per ton than he would originally for the wet coal. The benefit to the user would be: (1) A saving of freight on the water which would equal 16.7 cents a ton of dried coal, assuming $2.50 a ton freight rate. (2) The unloading cost would be reduced, as the coal would normally flow more readily from the car through the hopper bottoms. In freezing weather the saving would amount to many dollars if the frozen coal must be "mined" out of the car with hand labor. As this is almost impossible to evaluate on an average basis, we may assume that the wet coal could be treated with calcium chloride or rock salt sufficiently to prevent freezing. Assuming 15 lb of flake calcium chloride for freezeproofing one ton of wet coal, the cost for materials alone would be 20.1 cents for the chloride plus, say, 4.9 cents a ton for labor. (3) An excess of water in coal reduces the thermal efficiency of the steam generator in which it is burned. Each pound of water evaporated from the coal in the furnace is approximately one pound less evaporated in the boiler. More precisely, reducing the moisture content by 6.25 pct in the present example, the saving at the steam generator might well amount to approximately 4.5 cents per ton of dried coal used.

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* Assistant Director of Research, Bituminous Coal Research, Inc.

Improved fuel bed conditions and better coal handling as a result of lower moisture may well be worth more than this amount.

Neglecting other advantages of the lower moisture product, the above items alone total to 46.2 cents a ton, which is more in the winter season than the cost of drying.

The lament of the coal industry in postwar years was that the consumer did not seem willing to pay a fair return to the coal producer for the improved preparation of product. Whether this will continue to be the case in the future remains to be seen. However, producers who operate heat dryers should have little difficulty in disposing of coal at market prices should the demand fall off and strong competitive conditions return. Of course, we could discuss this part of the problem much longer than time will permit.

The paper under discussion shows that considerable information about the technical features of dryers, on which improvements must be based, are lacking, and furthermore that it is difficult to test existing installations to determine their performance. Although the writer agrees with the authors that thermal efficiency does not appear to be significant to the user of an existing dryer, yet the whole performance hinges on a reasonable efficiency. To an engineer interested in dryer performance and in improving the performance, the thermal efficiency and the entire heat balance take on considerable significance. Unless the heat balance is reasonable, the data are in serious question.

The authors have furnished the writer with average data for the three types of heat dryers tested in the survey based on only those tests which show reasonable heat balances. As shown in the following tabulation:

Heat Losses, Per Cent

Type of Dryer	Drainage Water	Evaporation	Exhaust Gases	Coal	Radiation and Unaccounted for (By Diff.)	Calculated Furnace Efficiency
Screen	1.5	33.1	13.8	18.2	33.4	87.8
Continuous Pallet	2.0	37.5	13.4	30.6	16.5	80.0
Rotary		49.0	16.4	19.2	15.4	82.1

The thermal efficiency is approximately 40 pct in the above examples as only the heat used in evaporating moisture is theoretically useful. Roughly 15 pct of the heat is lost in and about the furnace before the hot gases reach the dryer. The sensible heat gained by the coal averaged 25 pct and the exhaust gases carried out 15 pct. The balance, 5 pct, may be considered radiation and unaccounted-for losses from the dryer itself. These values are rough and approximate and will probably be subject to change when greater refinement in the measurement can be justified.

As the art of drying coal advances, more attention will be given to the techniques for testing dryers and for measuring their mechanical and thermal performance. The dryer designs appear to be based almost entirely on empirical results and much could be learned about basic factors in evaporating moisture from coal in streams of heated air. It is to be hoped that all those who plan to conduct tests on coal dryers or to measure basic data will keep in touch with the authors or with the sponsor that the art may be advanced most systematically and with maximum benefit to the entire industry.

M. G. Driessen*—May I start by congratulating Mr. Lyons and my friend Mr. Richardson for their excellent paper.

The principles of drying are generally the same for all the different apparatus, that is, heated air flows along the coal particles, and there is a heat transfer from the air to the coal or to the water adhering to the coal.

The heat transfer can be calculated as soon as the temperature difference between air and coal, the total surface of the coal particles and the heat transfer coefficient are known. Basically this is the same as the heat transfer in a boiler where the flue gases will convey their heat to the pipe bundles.

Osborne Reynolds in 1874 pointed out in a paper published by the Literary and Philosophical Society of Manchester, that heat transfer and the resistance or pressure loss of the gas are closely related, and this feature has been taken up by Professor F. K. Th. van Iterson[5] who shaped the law discovered by Reynolds into engineering formulas. He infers that the gas when in contact with the solid is relative to the solid at rest, and will assume the temperature of the solid. Because of the turbulent motion these gas molecules are then removed from the surface of the solid in little curls and eddies and conveyed in the gas flow.

In this way it is possible to calculate the heat transfer from the air friction by the following method:

If the force, which presses the gas or air through the many channels between the coal particles is called W:

$$W = pf \qquad [1]$$

Where p = pressure drop of the gas.
f = cross section of the channels in one plane.

According to the law of momentum this force will be equal to the increase of momentum of the mass M which is accelerated per second

$$W = Mv$$

v = final air velocity.

The weight of the mass M equalling

$$Mg = \frac{W}{v} g$$

will be cooled from the average temperature of the air to the temperature of the coal particles. If t is the temperature decrease of the air and C_p the specific heat of the air, the following amount of heat units (calories) will be conveyed to the coal particles.

$$\frac{W}{v} g\, C_p t$$

If F is the total surface of the coal particles to be dried we find for the heat transfer coefficient k':

$$k' = \frac{W}{v}\frac{C_p g}{F} \text{ cal per sec per } m^2 \text{ per degrees C}$$

or inserting Eq 1 and transferring to cal per h per m^2 per degrees C, we find

$$k = \frac{pf}{v}\frac{C_p g}{F} 3600 \text{ cal per sec per } m^2 \text{ per degrees C.}$$

By this simple deduction we find that there is a relation between pressure drop of the air, because of surface friction, and the heat transfer of the air to the coal particles. In many cases the pressure drop will be proportional to v^2 or to a power of v somewhat less than 2 (for smooth surfaces about 1.75).

Then the heat transfer coefficient will be proportional to v or to a power somewhat less than 1. In the above mentioned paper Professor van Iterson has derived the equation for flow through and along pipes, but the same principles are valid for a gas flow between coal particles, so that in many cases it will be sufficient to measure the pressure drop, and to derive the heat transfer from the pressure drop in the above indicated way.

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[5] F. K. Th. van Iterson: Warmte-overdracht van vaste lichamen op turbulent Stroomende vloeistoffen (Heat transfer of solids to turbulent flowing Fluids). *De Ingenieur* (April 24 1926).

I would like to ask Mr. Lyons if he found this relationship between heat transfer coefficient and pressure drop in his experiments.

O. R. Lyons (authors' reply)—I do not believe that the same relationships exist in a coal dryer as exist in a boiler.

In a coal dryer there is a different method of heat transfer in that the hot gases must pass over and through the coal rather than be transferred through a metal wall. The size of the coal, the amount of water contained in the coal, and the moisture content desired in the final product introduce variables not found in a boiler and exert considerable influence on the method and amount of heat transfer.

In other words, it is more difficult to transfer heat from a gas to a number of solid particles if the particles are not being turned to expose new surfaces, if the particles are relatively fine rather than coarse, and if the time of contact is very short.

It was difficult at some installations to tell whether the coal was being agitated sufficiently to turn it over continually or whether it was going through the dryer without any change in the position of the particles relative to each other. Even when it was evident that some particles were being turned and exposed, it was not possible to determine if every particle was being exposed.

Since this work was intended to be a survey, it was not possible to investigate what one might call the physical characteristics of the individual dryers in any great detail and therefore we did not attempt to determine if the relationship mentioned by Dr. Driessen actually exists in thermal coal dryers.

Recovery of Resin from Utah Coal

By Ernest Klepetko,* Member AIME

(New York Meeting, March 1947)

A notable amount of fossil resin exists in many of the bituminous coal beds of Utah. The upper part of these show a marked concentration of resin, which occurs primarily in the fracture seams. In general, these seams are less than ⅛ in. thick, but it is not uncommon to find coal with areas as thick as ⅜ in.

The resin is of the hard kauri variety, with coloration varying from deep blood red to lemon yellow and almost colorless. There is also an almost black variety, which is not readily soluble in the same solvents as the lighter colored resins.

These resins are valuable for many industrial purposes. When refined, they are suitable for uses similar to those for which lac resins are usually employed. They are particularly desirable for preparing dielectric varnishes. They are also desirable in preparing printing inks, waterproofing agents, special synthetic rubbers and rubber cements. They are especially suitable as an admixture in synthetic rubbers and cements, as they supply the essential tackiness the synthetics lack.

The following chemical trade characteristics have been found by Inter-Chemical Co. research chemists in the refined resins:

Specific gravity (melted).......	1.03 –1.06
Acid number..................	6–8
Softening point, mercury method	160°–165°C
Iodine valus (Wijs)............	140–150
Refractive index..............	1.544
Color (Hellige, 25 pct toluol)...	14
Molecular weight, average......	732
Purity, 99 + pct pure hydrocarbon	

"Unaffected by alkalies and resistant to one hour fusion with solid potassium hydroxide. Completely soluble in aliphatic and aromatic hydrocarbons. Insoluble in alcohols, ethyl acetate and other commonly used alcohols and esters. Compatible with natural and synthetic rubbers, chlorinated and cyclized rubbers, waxes, vegetable oils, mineral oils, rosin, ester gum, phenolic and modified phenolic resins and other natural and synthetic resins of like solvent solubility."*

The recovery and refining of these fossil resins has been the subject of much research work and has intrigued many investigators. Broadly speaking, the problem involves separation of the resin from the coal into a very high-grade concentrate, and subsequent treatment of this concentrate to produce an acceptable refined resin for its various usages. Obviously it is also desirable to recover the residual coal without excessive size degradation.

The late W. D. Green, concentration engineer for Combined Metals Reduction Co., obtained U. S. Patent No. 1773997 for the recovery of resin from coal through

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* Metallurgical Manager, Combined Metals Reduction Co., Stockton, Utah.

* Inter-Chemical Co.

the application of froth flotation. His patent also covers the use of various flotation reagents, such as higher alcohols, turpentine, cresol, pine oil and potash alum. This patent anticipates using the potash alum in the cleaner circuit to improve concentrate grade. It appears that a 30- to 40-mesh grind was considered most suitable.

As a result of this acceptable resin flotation procedure, arrangements were made with the International Smelting and Refining Co. to utilize its pilot plant at Tooele, Utah, for a more thorough investigation. It was also desired to produce large batches of resin concentrates for testing by possible consumers.

In these pilot-plant flotation tests, very satisfactory recoveries and concentrate grades were obtained. However, a standard pilot-plant ball mill and classifier was used. This resulted in the production of rather finely ground feed with attendant excessively fine concentrate production. A screen analysis indicated only 0.25 pct plus 100-mesh, 9.40 pct plus 200-mesh and 90.33 pct minus 200-mesh.

Samples of this concentrate were submitted to an eastern research laboratory for study. Its rather pertinent observation follows:

> The manner of separating the resin from the concentrate presents a problem which will require considerable further study. Separation by solvent means is the more desirable, since the resin is unaffected in any way by such treatment. Practically, however, the difficulty encountered in so doing concerns the separation of the insoluble portion from the resin-solvent mixture. The bulk of the insoluble matter, apparently carbonaceous in character as previously stated, is present in such a fine state that it will not settle on standing and instantly plugs up any satisfactory filter media.

The foregoing conditions indicated the necessity of devising means for improving the physical quality of the flotation concentrates to assure their acceptability for further refining.

Because of the outbreak of World War II, all research work was discontinued until the fall of 1942. At that time, it appeared that the production of a high-grade resin concentrate, from which a refined resin could be produced, might be of considerable value to the war effort. Accordingly a new research program was initiated by the Metallurgical Staff of Combined Metals Reduction Company.

SMALL-SCALE PILOT-PLANT INVESTIGATION

Samples of large lots from the coal seams of the Deer Creek and Salina Canyon districts were tested for concentrate yield by the Green flotation method, and the following results were obtained:

PLACE	RESIN CONCENTRATE YIELD, PCT OF FEED WEIGHT
Salina Canyon No. 1	5.96
Salina Canyon No. 2	4.36
Deer Creek: 3-in. lump	4.01
Stock pile	4.60
Screenings	6.10

Concentrates contained 75 to 80 pct soluble resin.

Many new features of the procedure developed for the flotation of the resin and its subsequent refining are regarded as patentable and applications have been filed with the U. S. Patent Office.

Coal Preparation

A careful examination of numerous coal samples from several mines disclosed that the resin always occurs in fracture planes of various thicknesses. Green's observations, that the finest resin size in these planes is on the order of 40-mesh, were confirmed. However, it was determined that resin is far more friable than coal. A simple laboratory test demonstrated that a pure resin particle as large as 6-mesh could be shattered under thumb-nail pressure, whereas heavy pressure in a pestle and mortar was required to break pieces of coal of similar size. Obviously the value of the coal tailing would be greatly enhanced if fine grinding could be avoided. Furthermore, operating costs in filtering and drying final tailings for sale would be lower.

To establish the minimum size required for satisfactory flotation, three identical 500-lb batches of coal were carefully reduced by graded crushing to pass through 20-mesh, 10-mesh and 6-mesh, respectively. Each batch was floated separately in a pilot-plant eight-cell mechanical machine. Substantially the same recovery of resin was obtained from each batch. Tests also made in a laboratory flotation cell on riffled samples from these batches, as well as on a minus 40-mesh special sample, proved that resin recoveries were practically identical. The quality of the concentrates from the coarser sizes was definitely superior. These tests were accepted as conclusive proof that a 6-mesh material would be desirable for production of a clean flotation concentrate. It was evident that every effort should be made to realize perfect graded crushing to eliminate presence of slimed coal in the concentrates.

Attrition in Standard-size Mechanical Cell

In order to obtain a large supply of resin concentrate for refining tests, a 7700-lb lot of coal was prepared by graded crushing to pass an 8-mesh screen; a 6-mesh of the desired size was not procurable. The entire lot was floated in batches in a standard 36-in. mechanically agitated single-cell flotation machine.

Table 1 shows the screen analysis of the various flotation materials.

The excessive attrition that occurred as a result of the mechanical agitation is well shown by the screen analyses. It is to be noted that whereas the original feed contained only 16.9 pct minus 100-mesh, the minus 100-mesh in the tailing increased to 62.1 pct, and that 93.7 pct of the concentrate was through 100-mesh.

TABLE 1—*Screen Analysis of Flotation Materials*

CUMULATIVE PER CENT

Mesh	Feed	Tailing	Concentrate
+ 8	1.1	0.1	
+ 10	17.9	1.1	
+ 20	41.5	3.6	
+ 48	71.7	10.7	0.3
+ 65	77.7	16.9	1.9
+100	83.1	37.9	6.3
−100	16.9	62.1	93.7
Total	100.0	100.0	100.0

The results of this test suggested use of a flotation cell that would provide ample aeration without excessive agitation; for instance, the old Callow blanket cell, the McIntosh machine or the airlift type of Forrester. Laboratory units of all three were tried. Each produced much cleaner and coarser concentrates than were obtainable with mechanically agitated machines. The airlift type gave less trouble with the coarse feeds being treated. It was found quite feasible to satisfactorily handle minus 6-mesh feed in a cyclic flotation circuit in specially designed airlift cells.

To compare the attrition characteristics of the unit with the aforementioned results obtained through the 36-in. mechanical cell, operating on minus 8-mesh feed, a large batch of identical coal was crushed to pass 20-mesh for flotation in a larger airlift cell. Thus, in order to exaggerate the conditions, the comparison was made on minus 8-mesh feed for the mechanical unit and minus 20-mesh for the airlift cell. Table 2 shows significantly the difference in resin disintegration in the two types of machines.

It is to be observed from the feed screen analyses that excessive fines are not produced with graded crushing even to minus 20-mesh. However, the concentrate screen analyses definitely prove how friable the resin is, as well as how much disintegration is caused by use of a mechanically agitated machine; that is, in spite of the fact that the airlift unit was operated on very much finer feed, the concentrate obtained therefrom was only 54.4 pct minus 100-mesh, as compared with 93.7 pct from the mechanical machine.

TABLE 2—*Relative Screen Analyses*
CUMULATIVE PER CENT

Mesh	Feed		Concentrate	
	Mechanical Cell	Airlift Cell	Mechanical Cell	Airlift Cell
+ 20...	41.5	14.7		3.8
+ 48...	71.7	58.0	0.3	21.7
+ 65...	77.5	70.5	1.9	32.5
+ 100...	83.1	79.3	6.3	45.6
− 100...	16.9	20.7	93.7	54.4
Total....	100.0	100.0	100.0	100.0

As a result of these investigations, it was concluded that the preparation of the feed material to the flotation cells, as well as the type of flotation machine, was of major importance in the production of acceptable resin concentrates.

The essential objective in the research program had been to determine methods whereby satisfactory flotation results could be maintained with simultaneous production of an acceptable grade and quality of concentrate. Concentrates had been produced, which, when treated with suitable solvents, gave solubilities of 70 to 80 pct. No serious difficulties were encountered in separating the resinous liquors from the inert solids, either by settlement or by filtration.

LARGE PILOT-PLANT OPERATION

Coal Preparation

The results of the preliminary investigation were regarded as sufficiently satisfactory to justify the construction of a pilot plant capable of handling approximately one ton per hour of coal. To make possible the treatment of any size of coal, a small screening and crushing unit was erected in a part of the old Combined Metals Reduction Co. leaching plant at Bauer (Fig 1).

The preliminary investigation indicated the desirability of reducing all coal, by graded crushing, to not finer than minus ⅜ in. and floating only the minus 6-mesh product obtained therefrom. This would involve the flotation of about 66 pct of the original coal.

All coal received either by truck or by car lots was delivered to an unloading ramp, which was served by an electrically operated drag slusher. This deposited the coal into a feed hopper and thence to a slow-moving feeder-conveyor belt. The belt-conveyor drive mechanism was geared to deliver approximately one ton per hour. The coal from the conveyor dropped to a small bucket elevator. This coal elevator discharged to a trommel 30 in. in diameter by 6 ft long, dressed with a ⅜-in. screen. The oversize passed by gravity through a small hammer mill and thence back to the boot of the elevator. The undersize from the trommel passed directly into a hopper-bottom, screw-feeder tank to which a small amount of water was added for preliminary wetting of the coal.

The screw feeder was set on an incline, so that all coal had to be wormed up through the water in order to assure its complete wetting. This wetted minus ⅜-in. material passed by gravity to a second trommel screen dressed with a very light 6-mesh, square-holed wire screen. Spray water, played on the outside of the lifting side of the screen, served to wash all minus 6-mesh material into a hopper, from whence it flowed by gravity to an agitated flotation feed tank. The minus ⅜-in. plus 6-mesh material was sluiced directly to a pump tank, which also served for rejection of the final flotation tailings. Since this coal contains less than one per cent recoverable resin, it is not regarded as economical to crush and float the material. Moreover, it is a desirably sized material to remix with the minus 6-mesh flotation tailing to produce an excellent stoker feed. This minus ⅜-in. plus 6-mesh material amounts to approximately 34 pct of the preparation-plant feed.

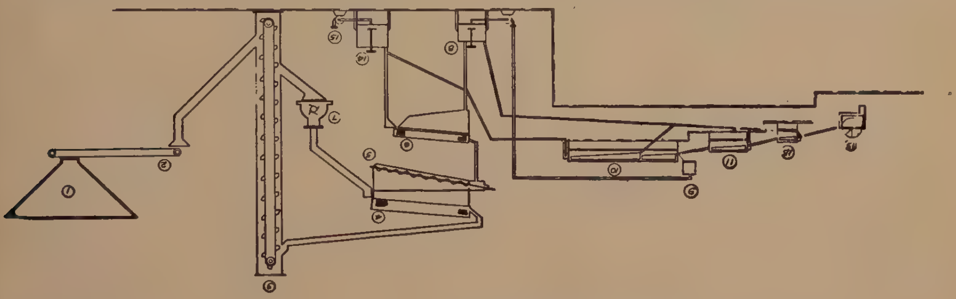

FIG 1—COAL-TREATMENT PILOT PLANT.

1. Hopper.
2. Feeder belt.
3. Elevator.
4. ⅜-in. trommel screen.
5. Wetting screw.
6. 6-Mesh trommel screen.
7. Hammer mill.
8. Flotation-feed pump tank.
9. Surge tank.
10. Flotation rougher.
11. First cleaner.
12. Second cleaner.
13. Vacuum leaf filter.
14. Tail-mixing pump tank.
15. Pump to dewatering.

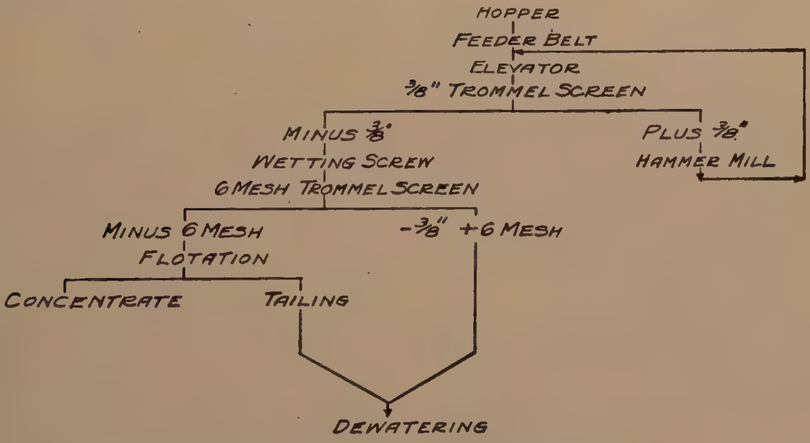

FIG 1*a*—FLOWSHEET COAL-TREATMENT PILOT PLANT.

Flotation Plant

The mixing tank that received the minus 6-mesh material, and which fed the flotation-feed pump, actually served also as the receiving tank for return flotation middlings and cleaner tails. Thus the flotation plant was unusual, in that only one pump served to circulate all materials through the flotation circuit.

As a result of our experiments with airlift flotation machines on the original small-scale pilot plant, it was decided to retain this type of machine for the larger unit.

The flotation machines consisted of one 14-ft long rougher, one 4-ft primary cleaner and one 2-ft final cleaner, all of the airlift type. The flotation boxes were 4 ft wide and 4 ft deep at the overflow lip on all cells. They were built entirely of ½-in. resin pressed plywood nailed inside a 2 by 4 framing and the sides of the tanks were hoppered on a 45° angle. The first 6 ft of the rougher delivered its concentrates to the first cleaner and obviously the concentrates from this unit were recleaned in the second machine. The middlings and both cleaner tails returned by gravity to the mixing tank of the crusher plant. Air was supplied by a Roots blower, which delivered 1100 cu ft of air per minute to the cells.

The usual method of connecting the roughers to the cleaners was dispensed with in favor of a more compact arrangement; that is, the first cleaner was set below but adjacent to the feed end of the rougher, so that the concentrates had a minimum distance to flow for retreatment. The second cleaner followed the primary cleaner in the usual manner. Thus, the rougher-concentrate launders sloped toward the feed end of the rougher instead of the usual reverse flow toward the tail end of the rougher circuit; thereby shortening the concentrate launders.

Special baffles in the airlift boxes were provided to assure breaking up of the bubbles for the production of a much finer bubble than ordinarily is found in this type of machine. The airlift column was only 6 in. wide, to assure maximum circulation of the unusually coarse feed. No trouble with sanding-in due to large coal was encountered. To assure continuous flow of the heavy material through the machine, a large spigot ran constantly from the bottom of the rougher cell. Pulp levels in the roughers and cleaners were maintained by the usual weir strips.

Over the complete period of its operation, more than 1500 tons of coal from various sources were treated through this plant. Under ideal operating conditions, on a three-shift operation, about 11 tons of coal per shift could be handled. However, because of inability to obtain labor, the average tonnage per shift on a one or two-shift basis was about 6 tons, of which approximately 66 pct actually passed through the flotation circuit, the plus 6-mesh material being of course rejected from the screening circuit.

It was found that pulp densities in excess of 15 pct solids gave considerable trouble in producing concentrates of the desired higher quality; about 11 to 12 pct gave the best results. While this density may appear low to the average flotation operator, accustomed to treating sulphide ores, it should be remembered that the specific gravity of coal is on the order of 1.6. Accordingly the bulk of solids in the machine actually approaches that common to the flotation of sulphide ores. In order to produce the cleanest concentrates possible, it was found that solids in the No. 1 cleaner should preferably be kept under 8 pct and in the No. 2 cleaner under 5 pct.

The concentrates were filtered on three tray filters. Because of the low specific gravity of the concentrates, less than 1.2, what would appear to be a dry concentrate for a metal mill contained from 22 to 26 pct moisture.

Many different reagents were tried in the plant, and satisfactory results were obtained when the total reagent combination did not exceed 0.4 lb per ton. Overoiling invariably brought up excessive coal into the rougher concentrates, which was almost impossible to reject in the cleaners. Accordingly great caution was used to control the reagents, and it was found most desirable to add a greater part of the frother into the middle of the rougher machine. A small amount of frother, such as amyl acetate, was fed at the head end to sustain a lively froth condition on the cleaners.

Concentrate grade was determined on the basis of resin solubility in the solvent found most desirable for subsequent refining. In general, it can be stated that the grade of the concentrates was never less than 70 pct, and more often nearly 80 pct soluble. All recoveries were calculated in percentage of coal weight, rather than of recovery of resin. This was done primarily because of the difficulty of determining the resin content of coal with solvents, since coal tar and other soluble ingredients complicate such a determination. However, laboratory flotation tests carefully made on daily feed samples crushed to pass through 10-mesh invariably checked quite closely the plant results. Since the plant was being operated to produce the greatest amount of acceptable concentrate, rather than to attempt to force recovery at the expense of quality of concentrate, the method of determining acceptable results was found to be quite satisfactory.

TABLE 3—*Summary of Coal Flotation-plant Data*

Date, 1944	8/8–8/12 (incl)	8/14–8/18 (incl)	8/21–8/25 (incl)	Total or Average
Number of shifts	10	13	14	37
Screening plant:				
Coal feed, dry tons	70.67	99.35	105.22	275.24
Tons per shift	7.07	7.64	7.70	7.47
Reject plus 6-mesh:				
Percentage of feed weight	33.9	34.0	35.2	34.4
Flotation plant:				
Feed, 6-mesh	46.64	65.37	68.28	180.29
Tons per hour	0.60	0.736	0.649	0.662
Solids, pct	11.07	11.4	11.2	11.22
Concentrates:				
Solids, pct	11.2	10.8	11.8	11.3
Filter cake, pct H_2O	25.1	26.4	26.7	26.1
Dry concts., lb	4000	9664	8695	22,359
Cleaner tails:				
Solids, pct:				
No. 1	4.0	5.8	6.6	5.47
No. 2	1.8	3.2	4.2	3.07
Reagents, lb per ton	0.455	0.453	0.397	0.435
Concentrate recovery, pct of coal weight	4.30	7.37	6.45	6.04

The data in Table 3 summarize plant results for the aforementioned treatment of something over 275 tons of coal screenings, or so-called "bug dust," obtained from one of the Utah coal mines.

These typical data covering the operation of the plant on 275 tons of coal feed show that 34.4 pct of the feed weight was rejected as a minus ⅜-in. product. From the 180 tons minus 6-mesh material, which went to the flotation plant, there were produced 22,359 lb of dry concentrates; or there was an average yield of 6.04 pct of the weight of the original coal in the form of resin concentrate.

Since proper sizing is vital to optimum flotation results, the average screen analyses covering a typical operating period are given in Table 4.

TABLE 4—*Average Screen Analyses in Typical Operating Period*

Mesh	Original Coal, Pct		Flotation Feed, Pct		Flotation Tails, Pct		Flotation Concentrate, Pct	
	On	Cum.	On	Cum.	On	Cum.	On	Cum.
+ 10	29.7		11.6		11.6			
+ 20	26.8	56.5	29.4	41.0	33.5	45.1	3.9	
+ 30	14.6	71.1	20.2	61.2	20.6	65.7	10.6	14.5
+ 40	3.2	74.3	4.4	65.6	3.7	69.4	4.4	18.9
+ 60	12.2	86.5	13.6	79.2	14.4	83.8	22.6	41.5
+ 80	4.8	91.3	5.9	85.1	5.4	89.2	16.0	57.5
+100	0.4	91.7	0.7	85.8	0.5	89.7	3.7	61.2
−100	8.3	8.3	14.2	14.2	10.3	10.3	38.8	38.8

It is again pointed out that the objective in operating this pilot plant was to produce a large supply of concentrates for further study in a pilot refinery. At the same time it was desirable to determine the quality of the coal reject that would be produced. No difficulty was encountered in utilizing this coal, either in our own plant stokers or at other industrial plants. As a matter of fact, the quality of the coal had been enhanced, since the removal of the resin actually increased the heat value in two ways. The resin has a value of approximately 8000 Btu, whereas Utah coals can be expected to have a Btu value of 13,000. Consequently, removal of 5 to 7 pct of the low-value resin obviously enhanced the value of the coal. The second source of improvement in the coal value was brought about by the elimination of slimy coal slate, rejected in a small settler, which in turn delivered its product to the coal cars. The coal tailing water overflowing the top of the settler carried away a great part of this slimed bony material.

The following characteristics of a car lot of coal show the type of tailings produced (dry basis):

Btu................	13,132
Ash, pct............	8.33
V.C.M. pct..........	42.59
Fixed carbon, pct....	49.08
Sulphur, pct.........	0.59

The comparative screen analyses of several shippers' coal screenings in Table 5 indicate that no degradation of the pilot-plant coal was evident.

TABLE 5—*Comparative Screen Analyses, Coal Screenings*

CUMULATIVE PER CENT

Mesh	CMR Co. Pilot Plant	U. S. Fuel Co.	Utah Fuel Co.	Independent Coal Co.
+ 4...	5.0	8.4	4.3	5.4
+ 10...	34.3	42.9	39.6	46.4
+ 20...	62.2	64.8	63.9	66.8
+ 48...	85.3	86.5	85.7	85.2
+100...	92.9	94.3	93.3	92.3
+200...	97.2	97.9	97.1	96.2
−200...	2.8	2.1	2.9	3.8
	100.0	100.0	100.0	100.0

RESIN REFINING

It was the original intention of Combined Metals Reduction Co. to sell concentrates directly to eastern concerns for final refining. A 12.5-ton test shipment of resin concentrates was shipped to an eastern chemical plant for refining tests but because of the many manipulation difficulties in attempting to process the material, it became evident that a refining process, as well as plant design, would have to be provided by the Combined Metals Reduction Company's staff.

The commercial flowsheet, as developed, involves steam hearth drying of the concentrates; continuous mechanical agitation leaching; separation of the liquor from the solids by pressure filtration with countercurrent washing; pressure clarification of the recovered liquor; partial distillation and recovery of the solvent from the liquor; final recovery of the resin and solvent through an electrically heated still, and melter and recovery of the residual solvent in the concentrate residue by hot-water treatment.

TABLE 6—*Characteristics of Solvents*

Solvent	Boiling Point, Deg C	Resin Solubility, Pct
Acetone....................	56.5	1.2
Amyl acetate..............	100.00	6.0
Methyl amyl alcohol........	64.6	Trace
Ethyl ether................	34.5	72.3
Benzol.....................	80.0	86.9
Carbon disulphide..........	46.3	61.1
Stoddards solvent..........	120–140	86.7
Petroleum ether (hexane)...	54–88	84.0

The initial investigation involved determination of the most satisfactory solvent for refining. Dissolving ability, filtering and settling characteristics and boiling point of the solvent were the principal factors involved. More than 50 different volatile solvents were tried in the laboratory. Table 6 shows the solubilities and boiling points of the various types.

Benzols and petroleum ethers proved the most desirable for all conditions imposed.

It was finally decided to concentrate all further work on hexane, one of the petroleum ethers, because of satisfactory solubilities, its specific gravity, and the rather desirable range of its boiling point. While benzol was regarded as also quite acceptable, its poisonous characteristics justified rejection as a desirable solvent.

The development of the flowsheet was a series of progressive steps. Small-scale leaching had indicated 2 hr at room temperature for dissolving, pressure filtration for separating the pregnant liquor from the solids and partial evaporation in a still. Accordingly, a pilot plant capable of producing from 400 to 600 lb per week on a one-shift basis was constructed. It should be remembered that this work was carried on during the war, when the procurement or construction of even the simplest type of apparatus was a considerable chore. For instance, the leaching tanks consisted of 110-gal steel drums set on end. In order to assure gas-tight leaching units, cylindrical disks welded to the shafts were rotated in a water pan welded to the top of the tank. While these were far from satisfactory, they did serve the purpose. In spite of the fact that more than 35,000 lb of refined resin was produced, we were fortunate in having no explosions or fires.

Filtration

It was thought that countercurrent washing and decantation could be accomplished in settling cones using Dorr diaphragm pumps to advance the mud from cone to cone. We were, however, sadly disillusioned as to the pumping of volatile liquids. The pumps would advance the material for a period of time; eventually all flow would stop, owing to vapor locks resulting from the vacuum created in the suction pipes. While it is believed that larger pumps and cones might possibly have worked satisfactorily, the small scale of the operation was so troublesome that this method was abandoned in favor of direct filtration.

All leaches were made in batches with 2 hr of mechanical agitation, with a solvent-concentrate ratio of 4 liters to 1000 grams. Settling tests on this type of leach indicated a rate of about 5 in. per hr. Pressure filtration at 20 psi gave filter rates of 10 to 15 gal per sq ft per hr and production of ¼-in. wet cake. Vacuum filtration was not regarded as feasible in commercial operation because of flashing and plugging of the filter media caused by evaporation.

It should be stated at this time that the residue from a concentrate leach contains a great deal of sandy coal and insoluble dark resin, which has a very high settling rate in the pregnant liquors obtained. These rates rather baffled us at first. The explanation actually was simple, since the pregnant liquors seldom have a gravity of much over 0.752, whereas the rather coarse solids have a gravity of something on the order of 1.23. Gravities of rich liquors in hydrometallurgical operations are of course always nearer to 1.4; consequently our deductions had to be closely watched, as we were working with many conditions quite foreign to our normal problems of metallurgy.

Recovery of Resin from Liquors

The recovery of the resin from the pregnant liquors is theoretically quite simple, since evaporation of the solvent is the only requirement. Actually it became one of the major problems. Distillation of 65 pct of the solvent in a standard steam evaporator operated in closed circuit with a water-cooled tube condenser is quite readily accomplished. Since the hexane begins to distill at 54°C and finishes at 88°C, the temperatures involved are low. It was established that when the liquor reached a temperature of 65°C in continuous still operation about 65 pct of the hexane had been eliminated. The rather striking difficulty in the operation of the still was to accomplish this with a small amount of steam. The evaporator was built in our own shops from tanks and pipes that were available in the scrap pile of the plant, and the heating element was entirely too short for efficient operation. Even with 100 lb steam pressure, there was a marked insulating effect due to formation of gas bubbles upon the heating elements. The rate of hexane elimination per square foot of area per hour was exceptionally low, although well within economical limits. Approximately five times as much surface area was required as would normally be used in the operation of a still for hexane alone. Since it was not considered physically feasible to draw off the residual molasses-like liquor when evaporating more than 65 pct of the hexane, the subsequent treatment of this product was the subject of many proposals.

The resin itself has remarkable insulation qualities. Skin evaporation on drum evaporators or driers suggested itself. However, when a thin layer was applied to a heated sheet of metal, such as the surface of a drum drier, the evolution of hexane gas caused the formation of large bubbles, which formed a perfect insulator for further heat transference. To convey a picture of how excessive this bubble-forming characteristic was, the following illustration is submitted. If a ¼-in. layer of pre-evaporated liquor is evaporated in a regular household roasting pan in a steam drier, the hexane-free resin will have swelled to a depth of 6 in.

Since the hexane is completely volatile at 88°C, it appeared that the use of boiling water would solve the problem. A closed horizontal tank, provided with two agitators, gas outlet connections to a condenser and inlets for the pre-evaporated liquor, means for introducing steam and recirculation of hot water, was constructed. A heavy flow of hot water was circulated through this tank and a separate skimmer tank. Sufficient steam was passed into the tank water to maintain an 88°C temperature. A desired flow of pre-evaporated liquor flowed directly from the still into the agitation unit. Evaporation of the hexane occurred instantly when the liquor dropped into the hot water. This caused simultaneous liberation of the resin into spongelike particles, which flowed with the circulating water into the skimmer tank. This same procedure proved to be entirely satisfactory for treatment of the residual mud, for final hexane recovery, contained in this product.

Because of its porous structure and low specific gravity, the resin floated to the surface and was skimmed off and delivered to a tray filter. The excess hot water from the skimmer was recirculated to the resin precipitation tank. This procedure was technically satisfactory, but the recovered resin, even after filtration, contained over 50 pct moisture because of its cellular structure and low specific gravity. Centrifuging did not reduce the moisture below 45 pct and the ultimate drying of the recovered resin introduced a new problem.

A multiple-hearth, steam-heated McDougal type of drier was used satisfactorily. However, the steam consumption for eliminating so much moisture and the production of a very dusty product were disadvantageous. A further objection was the bulkiness of the dried material; viz, 20 lb per cubic foot. The unit used for this operation was built around an old leaching tank in the pilot-plant building. The unit consisted of six plate-steel hearths resting on steam coils. Ventilation to remove moisture was provided by a small suction fan. Temperatures, obtained from the 100 psi in the coils, ran as high as 120°C in the middle section of the chamber. Since the entire procedure of hot-water precipitation, filtering and drying was abandoned in favor of direct steam atomization, any further discussion of this method can be eliminated.

Atomizing Unit

The principle of utilizing live steam in the water-precipitation scheme suggested the idea of direct atomization with steam. The desired results imposed the production of a substantially dry product without the use of separate precipitation, filtering and drying steps. Since water and hexane are not miscible, separation of the condensed vapors offered no problem. It was decided therefore to proceed with the construction of an atomizing unit.

The atomizer was built from a former leach tank. A domed cover, with the desired outlets and openings for injection of the resin liquor, was fabricated to provide a gastight cover. Two steam-heated rabbled hearths served to collect and complete the drying of the atomized material. Side coils, shielded by a sheet-metal internal cylinder, furnished additional heating surface.

Five methods for atomizing the resin liquor into the chamber were anticipated in the design; namely, dropping the pre-evaporated liquor onto a high-speed rotating disk; vertical or side spraying of the liquor through a steam-liquor-mixing atomizer; atomizing the pregnant liquor into the chamber through one spray with independent injection of live steam; dropping a liquor stream across a steam-atomizing nozzle; and finally atomizing the liquor into the expansion chamber into which heat was supplied by recirculating a part of the hexane gas through steam-heated thermoliers.

Fig 2 illustrates the structural characteristics of the unit. The operating results by atomization were eminently satisfactory. The most desirable procedure for atomization was that of intersection of a stream of the liquor by cross flow with a steam jet. The atomizing results, when using live steam for heating, were a marked improvement over water precipitation. The recovered resin contained less than 2 pct moisture and owing to the presence of water vapor in the chamber, there were no explosion hazards. Since the unit was constructed from old equipment, it was not entirely gastight and unnecessary solvent losses occurred.

Although steam consumption appeared somewhat high, much heat could be conserved by passing the hot solvent-water-vapor gases, having a temperature of 88° to 96°C, through an interchanger against the advancing pregnant liquors that entered the pre-evaporation still.

It was found that the last traces of moisture, owing to the high water-vapor content within the chamber, remained in the final product. Moistures of 3 to 5 pct persisted. To accomplish elimination of this remaining water, an open steam-jacketed screw conveyor was installed to receive the last hearth discharge. Although having only a 2¾-in. diameter, 60-in. long screw, this conveyor reduced the moisture to approximately 2 pct. It was evident that a properly dimensioned conveyor would be quite satisfactory.

The procedure of spraying the pre-evaporated liquor into the spray chamber, heated by circulation of hexane gas, proved difficult. Excessive circulation of this gas through the steam-heated elements created considerable movement of fine resin through the heaters, with resulting insulation. Steam consumption per pound of resin recovered was exceptionally low but gas escape hazards were great.

Of the five atomizing methods tried, the most satisfactory results and simplest manipulation were those of gravity flowing the heavy liquor across a steam jet. This procedure is identical with that

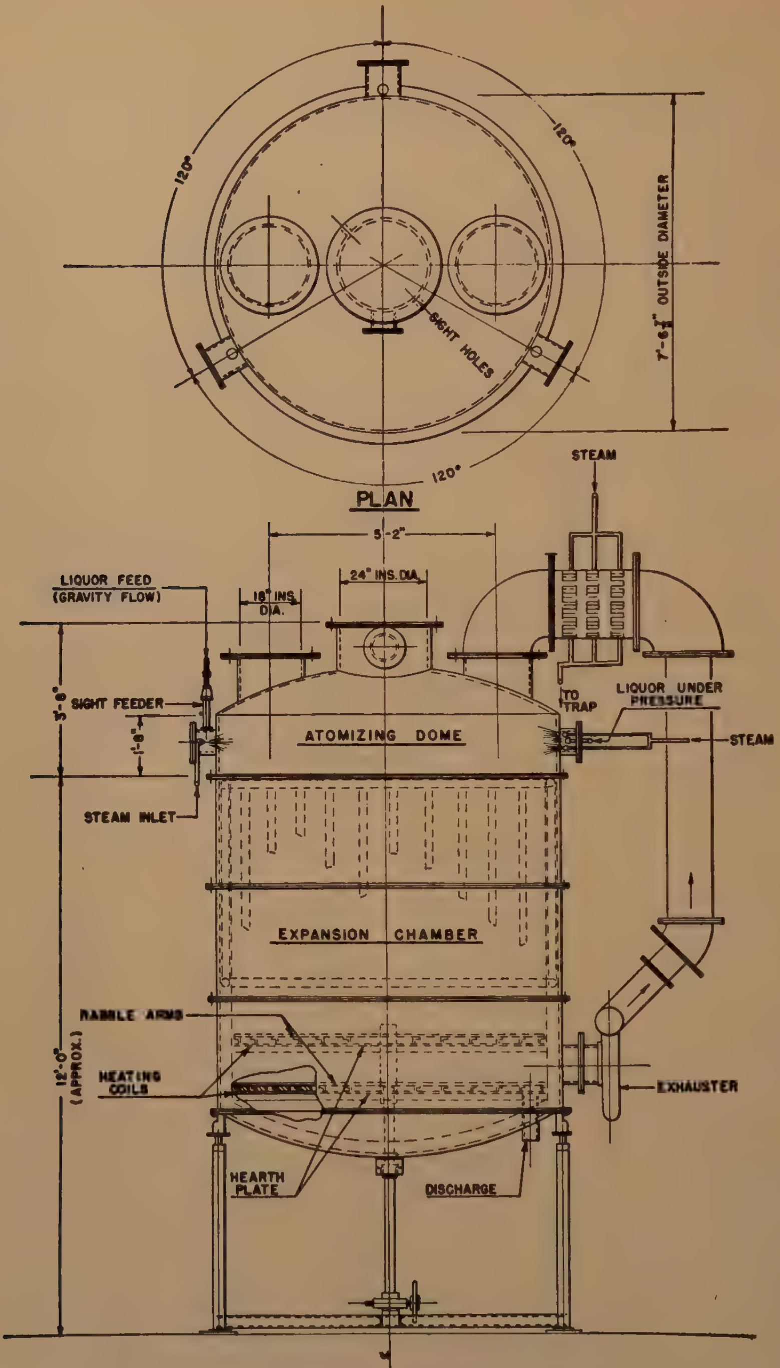

FIG 2—ATOMIZING UNIT.

employed in producing zinc dust for the electrolytic zinc process.

Pumping the pre-evaporated liquors to obtain desired atomizing pressures was a problem, though a piston type of pump with positive actuated valves would probably have been satisfactory. In any case, any one of the methods tried proved to be technically feasible.

It was intended to pass this atomized product into a pelletizer for final compacting. A manufacturer's test indicated that the pellets would weigh 40 lb per cubic foot. The pelletizing was not completely solved by machine manufacturers, and it appeared that certain modifications of the standard units would be necessary.

Melting for Shipment

Eastern consumers advised that a properly melted resin would be the most desirable product to handle. From the viewpoint of packaging and shipping, melting offered many advantages. Melted resin weighs approximately 80 lb per cubic foot. It was decided therefore to develop a method that would meet the technical requirements. Melting starts at 195°C and at 225°C the material is sufficiently mobile to permit casting into drums. It is particularly desirable that the material should not be kept at elevated temperatures for any great length of time, one hour being the upper limit. Therefore, continuous melting was indicated.

Because of the low melting characteristics of the resin, the possibility of increasing the density of the material by subjecting it to the heat of an infrared lamp was tried. A 14-ft 10-in. slow-moving belt, with feed hopper, over which were suspended thirty-two 250-watt lamps in two rows of 16 each provided a test unit. This equipment had a capacity of 32 lb per hr. The procedure was technically possible, but dropped in favor of the following more attractive melting procedure.

An electrically heated melting unit has been devised, which has proved to be eminently satisfactory. In a 2-ft diameter, 4 ft long melter, provided with two agitators, 200 lb per hr of resin has been melted in continuous runs of long duration. Fourteen 1500-watt strip heaters placed around the body of the carefully insulated shell sufficed to supply the necessary heat. A power consumption of much less than ¼ kw per pound of resin was required to accomplish the melting. The pre-evaporated liquor was diluted with a thinner before being added to the melter. This thinner, which has a vaporization temperature just below the initial melting point of the resin, serves as a vehicle to maintain plasticity until actual melting takes place. This vehicle, of course, is condensed and recirculated in the melter system. The melted resin weighs about 75 lb per cubic foot and therefore is eminently satisfactory for shipping.

COMMERCIAL RESIN REFINERY

A resin refinery is under construction at Bauer, Utah, and it is anticipated that it will be in operation before the close of 1947. This plant will have a capacity of one ton per hour of concentrates. Initial feed for the plant will be obtained from the Hiawatha coal washer of the United States Fuel Co. The amount of concentrates that can be obtained from this source has not as yet been determined and can hardly be fixed, since it is dependent on seasonal operations of the coal mine.

In order not to disturb the regular operations of the washer, a resin-recovery method had to be devised that could be carried on as a normal part of the plant operations.

The construction of a flotation plant, to augment the concentrate supply obtained from the Hiawatha washer, has been anticipated.

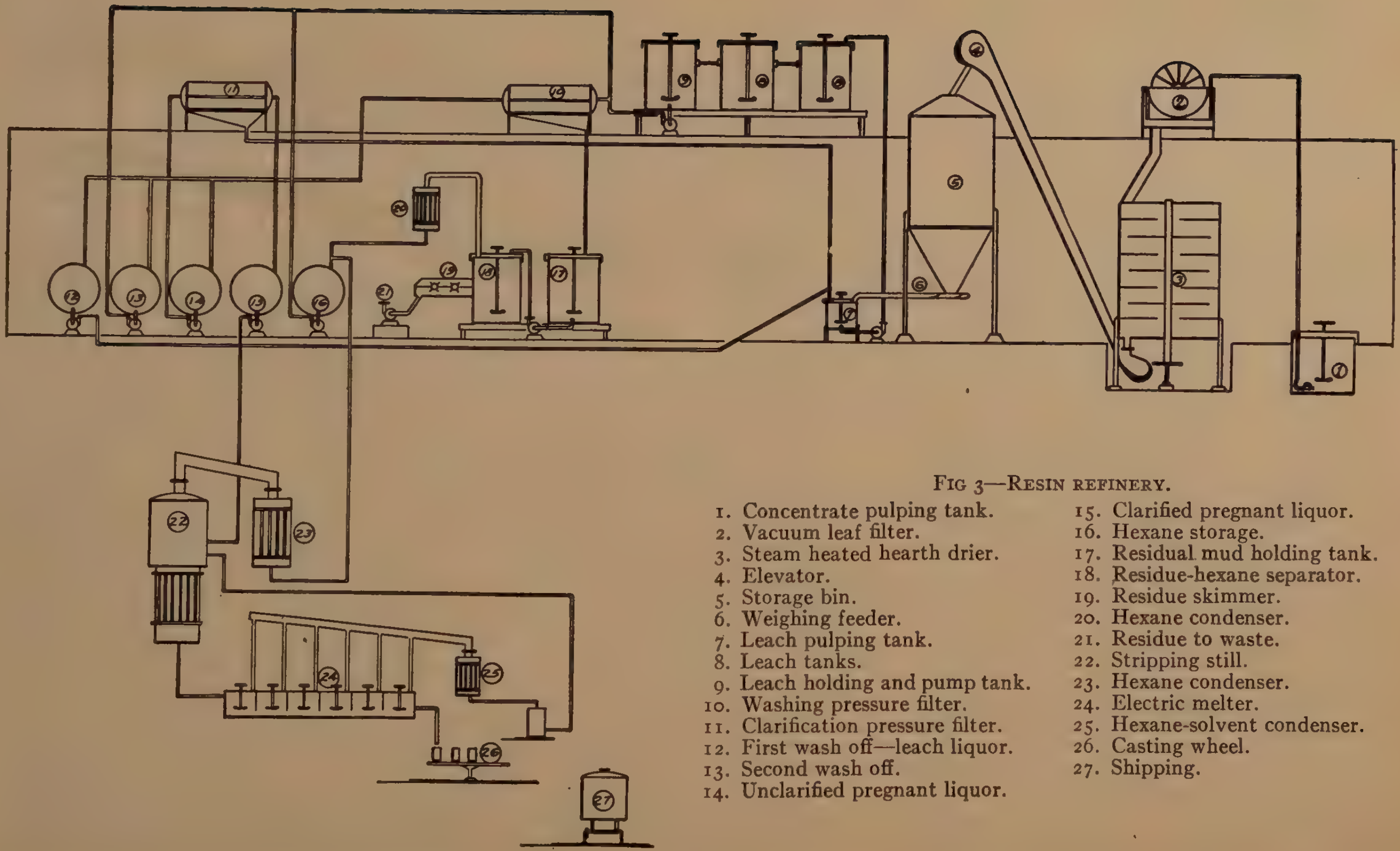

FIG 3—RESIN REFINERY.

1. Concentrate pulping tank.
2. Vacuum leaf filter.
3. Steam heated hearth drier.
4. Elevator.
5. Storage bin.
6. Weighing feeder.
7. Leach pulping tank.
8. Leach tanks.
9. Leach holding and pump tank.
10. Washing pressure filter.
11. Clarification pressure filter.
12. First wash off—leach liquor.
13. Second wash off.
14. Unclarified pregnant liquor.
15. Clarified pregnant liquor.
16. Hexane storage.
17. Residual mud holding tank.
18. Residue-hexane separator.
19. Residue skimmer.
20. Hexane condenser.
21. Residue to waste.
22. Stripping still.
23. Hexane condenser.
24. Electric melter.
25. Hexane-solvent condenser.
26. Casting wheel.
27. Shipping.

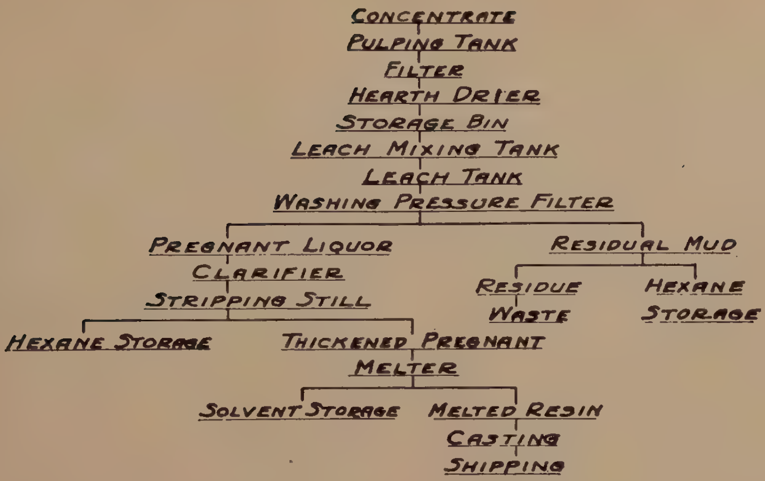

FIG 3a—FLOWSHEET OF RESIN REFINERY.

The flowsheet for the commercial refinery (Fig 3) indicates the equipment and material flow required for carrying out the process.

ACKNOWLEDGMENTS

I am deeply indebted to Mr. E. H. Snyder, President and General Manager of Combined Metals Reduction Company, for permission to present this paper. My appreciation goes to him for his sympathetic understanding of the problems which confronted us.

Due credit must be given to the late W. D. Green for his pioneer work in the recovery of resin concentrates from coal by flotation. I am also grateful to L. J. Hartzell, Philip Kaye and V. Earl Bell for their loyal support and untiring efforts in carrying out the research and development program.

DISCUSSION

(*J. Griffen presiding*)

L. C. McCabe[*]—This recovery of resins from coal is a very interesting development and it bears out some of the work that the Bureau has been doing along this very line. W. A. Selvig of the Bureau Station in Pittsburgh has done considerable work on resin extraction and has published an article recently on resin and wax recovery from coal. These recoveries come from the lower rank coals from the West and from the lignites of Texas, California and Arkansas. There have been very few waxes and resins found in the lignites of the Dakotas. They are essentially devoid of resins and waxes.

From the economic point of view, it is very interesting to recognize that we are developing here an industry that formerly did not exist or existed to a very limited extent in this country. Before the war the Germans exported to this country, if I recall correctly, something like 4,000 tons of waxes and resins a year, which sold at 60 to 70¢ a pound. This source of waxes now is in the Russian zone of Germany and is not available to us so I do think we have an opportunity of developing, in a small way, a technically important and, perhaps, a very economically important industry.

I think it will be limited to the coals of younger geologic age, that is the lignites and the subbituminous coals and the bituminous coals such as those of Utah. While resins and waxes are present in the carboniferous bituminous coals, the physical-chemical differences seem to disappear as the coals get higher in rank.

I think we shall, perhaps, see further development in the recovery of resins and waxes.

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[*] U. S. Bureau of Mines, Washington, D. C.

J. Griffen—The paper on the recovery of resin from Utah coal brought up a very interesting angle, on whether different types of flotation machines actually degrade the coal and in that way give a more difficult problem in filtering and a higher moisture content in the cake, or whether all that difficulty is simply due to the fact that it is froth, but the size consist has not particularly changed. Actually, in cleaning by flotation, you should tend to make a froth that is more coarse than the feed, because the finer fractions usually contain less coal and more refuse. But we know that a filter has lower capacity per square foot on flotation coal than it has on raw coal—that is, slurry from the raw coal not floated.

E. Klepetko (author's reply)—It is interesting to note that the Bureau of Mines Station at Pittsburgh has been working on resin extraction and I trust that we may expect, in time, a report of their investigations. We, in Utah, feel that perhaps we are making a further contribution to the mineral development industry of the United States. Our company has already constructed a commercial plant that has been in successful operation since the 17th of March, 1947.

Maintenance of a Coal Cleaning Plant

By Ralph M. Hunter,* Member AIME

(New York Meeting, February 1948)

Until recent years, maintenance of surface coal handling facilities was a relatively simple task. Equipment consisted principally of conveyors, screens and crushers of comparatively simple construction. Electrical equipment consisted of motors and starters, with little, if any, interlocking. In most cases failure of any component affected only the grades of coal loaded so that even a prolonged breakdown did not affect mine operation materially.

However, with the introduction of modern cleaning plants, the picture has changed. The equipment is much more complex and therefore more difficult to maintain. In many cases failure of any component of the plant prevents loading of coal. Even if provision is made to bypass, the quality of coal loaded is lowered, affecting markets. In the case of plants designed to permit full seam mining, marketable coal cannot be loaded without operation of the cleaning plant. Under these conditions it is apparent that considerable thought and effort toward better maintenance are justified.

In order to assure continuous, efficient and economical operation, maintenance should receive consideration, not only after the plant is in operation, but while it is being designed. It is hardly necessary to say that the designer should strive for sturdiness and wearing qualities. It is sometimes not realized, however, that even the best parts will sooner or later require replacement or repair. The designer should, therefore, give consideration to accessibility. Anyone who has had to do with cleaning plant maintenance is familiar with installations where several hours are required to get at a part that can be repaired or replaced in a few minutes, once it is accessible. As in most cases, experience is the best guide. The men who operate and maintain this equipment know the problems encountered. Whenever available, the opinions of these men should be sought and heeded.

As an example, we might consider the refuse elevator on a Chance Cone. Our company is now operating three of these cones, the first of which was installed in 1941, and the latest in 1946. Exact records are not available, but well over 50 pct of delays charged to the Chance Cones have been caused by the refuse elevator. This elevator receives refuse from the cone and carries it to a point above the water level of the cone, discharging onto a dewatering and desanding screen. Since the elevator operates under water and necessarily has close clearances in the lower boot, it tends to jam on large pieces of refuse or tramp iron. The chain, buckets and boot are also subjected to the action of water, sand and refuse, which accelerate normal wear. In case of jamming or chain-breakage it is necessary to drain the elevator and the cone so that even a minor fault in the elevator causes a considerable loss of time for the cone.

In our new plant, which will contain two Chance Cones, the refuse elevator has been eliminated. This was accomplished by raising the cone so that the refuse gates could discharge directly onto the dewater-

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* Rochester and Pittsburgh Coal Company, Indiana, Pa.

ing screen. Certain modifications in the water circulating system and an additional sand sump were also necessary. The initial investment is somewhat higher but we expect to be able to maintain better operation and reduce maintenance costs.

GENERAL LAYOUT

Maintenance should also be considered in the general layout of the plant. It is often possible, for example, to divide the equipment so that the plant can continue to operate at reduced capacity even though one element is out of service. In our fine coal plants, made up of Deister tables, these tables are divided into groups of six, each with its own distributors and settling tanks. Any group can be cut out and the plant operated at less than normal capacity.

In some cases, installation of spare units is justified. For example, centrifugal driers must be shut down periodically to replace baskets. In our fine coal plants we install an extra unit so that one unit is always standby.

PLANT CAPACITY

Consideration should also be given to plant capacity. Maintenance is simplified and improved if plant capacity allows some idle time each day. Preventive maintenance carried on daily allows equipment to be kept in such condition as to operate without delays. During the war we operated many of our plants on virtually a 24-hr day, 6 days a week schedule. With only Sundays available, a backlog of routine repairs was quickly built up. This backlog was constantly threatening to overwhelm the maintenance crew. The present wage scale, with penalties for sixth and seventh day work, will justify an additional investment in increased capacity.

ORGANIZATION

In order to properly care for maintenance an organization is necessary. This organization may be simple or elaborate depending on the size of the plant and its complexity. Any organization, however, should provide for fixing responsibility, for inspection, for regular repairs, for handling emergencies, and for keeping maintenance records.

It is my own opinion that responsibility for operation and maintenance should be combined. The operating head should know more about the current condition of equipment than anyone else and he is also in a position to control operations to prevent abuse of equipment. Maintenance must be performed with the least interference to production. In fact, these two functions are interdependent.

Regular, daily inspection will catch many breakdowns before they occur. This is one argument against a central oiling system and, to my mind, a valid one. The oiler, properly instructed and trained, is about the best inspector to be found, and in his regular rounds he visits every piece of machinery and is able to note variations from normal.

In addition to the daily inspection of the oiler, the foreman or someone delegated by him should make regular inspections while the plant is in operation. Excessive heat, excessive noise, oil leaks, and other abnormalities should be noted for investigation. In addition, housekeeping, condition of stairs and walkways, condition of lighting and wiring should be observed. Inaccessible equipment, such as flexible couplings, gears, pumps, submerged parts of jigs, drive chains and belts should be inspected regularly while not in operation. Air gaps of motors, especially sleeve bearing motors, should be gauged regularly.

One great enemy of cleaning plant equipment, both electrical and mechanical, is dirt. Controllers, contacts, limit switches, and the like are especially susceptible to dust. Regular attention with an air hose, and frequent visual inspection will keep these parts in working order.

A regular repair crew, working off shift, is usually necessary. This crew should be made up of skilled men capable of burning, welding, fitting bearings, or handling any of the various jobs bound to arise. In most plants a crew of about four men works out to the best advantage. With such a crew working regularly, it is possible to replace worn parts, such as chutes, wearing plates, and the like, before they fail. Inspections of equipment not accessible when the plant is operating can also be handled by this crew. While the repair crew should be separate from the operating personnel, they should be under the direction of the operating foreman and receive their instructions from him. Above all, the repair crew should not be considered a place to dump inefficient personnel. Rather, assignment to the repair crew should be considered a promotion.

Even with the best program of preventive maintenance, breakdowns on shift will occur. It is necessary, therefore, to provide for these emergencies. The regular operating crew will in most cases be competent to handle these emergencies if they have been properly instructed. A large plant will usually justify one or more utility men who have been trained to locate and correct troubles promptly. With a proper program of preventive maintenance, on-shift troubles are likely to be electrical, improper operation of interlocking or motor or control failures. Proper training and instruction are necessary to locate these troubles promptly and usually the big problem is locating trouble.

In our case, we make the electrical installations with our own crew and it is possible to assign a maintenance man to work on the construction. This gives him a chance to become acquainted with the installation in detail.

VALUE OF RECORDS

To most maintenance men records are a nuisance, to put it mildly. However, records are an essential part of proper maintenance. I hesitate to expand on this point because our own system of records is not complete. However, to be useful, records should be easy to keep, should give the necessary information, and should be used in planning maintenance. The ideal system would allow replacements or repair to be made just before failure. While this objective is probably not attainable, it is possible to schedule overhaul of major equipment so as to fit in with production schedules. Proper records will aid in such scheduling.

SUFFICIENT SPARES

Maintenance records are also of value in determining the extent and composition of the spare parts and repair supplies inventory. The problem of inventory is difficult and calls for careful analysis. Excessive inventory ties up capital and storage space and is therefore undesirable. On the other hand, lack of a part may shut down the plant. In a new type of plant, it is likely that more spare parts than necessary will be carried but operating experience will show the safe minimum.

In this instance also, proper design can help. A little care in selecting speed reducers, motors, drive chains and sprockets, and other parts will allow relatively few spares to protect the important units of the plant. Especially in these times of long deliveries it would seem advisable to carry enough spare parts to replace at least the most important units. Even with the most careful planning, however, complete replacement units are likely to build up inventory to unreasonable size. One solution to this problem is advance planning for breakdowns. Obviously, a 15 hp motor and reducer will handle the load usually carried by a 5 or 10 hp unit. Often, space will allow the larger unit to be installed. A little advanced planning will provide adapter plates, coupling halves, and other material necessary to adapt the available unit to the necessary application. While these adapters can be made up in advance, having them available when needed is another matter.

We have had some success in accomplishing this by storing adapters near their intended use, hanging them on brackets or racks.

At present we are operating two modern cleaning plants with a third under construction. Many of the drive units in the three plants will be duplicates and in the three plants a total of approximately 250 motors is involved. A plan has been proposed and is being developed to pool spares for the three plants. This plan has some very obvious advantages, principally reduction of investment in inventory. It has some equally obvious disadvantages, chief of which is the problem of quickly locating and delivering the needed spare unit and controlling repair and return to stock of damaged units. We feel that the plan has merit and are now exploring the possibility of a coding system to identify the unit to be replaced and the spare unit. One danger to be guarded against in setting up such a system is to find we are running a system instead of maintaining a cleaning plant.

Assuming that motors are properly loaded originally, periodic load checks of motors will often indicate troubles before they become serious. These checks are quickly made with a clip-on ammeter and, if made regularly and recorded, are invaluable in catching potential trouble both mechanical and electrical before it happens.

Summary

To sum up, good maintenance in a cleaning plant requires:

1. Careful design from a maintenance point of view.

2. Definitely assigned responsibility for maintenance.

3. Regular, systematic inspection to locate troubles before they become breakdowns.

4. A well organized and equipped maintenance crew, with time to make repairs when they are needed.

5. Sufficient records to spot slowly developing troubles and to form a basis for long term maintenance planning.

6. Sufficient spares to replace quickly important units.

In addition to the above, a cleaning plant foreman needs a sixth sense to spot trouble before it occurs. But until a practical crystal ball is developed, careful inspection, study of past experience and systematic checking will have to do.

DISCUSSION

(*T. W. Guy and J. Griffen presiding*)

T. W. Guy—I think Mr. Hunter has called attention to a great many points that are important in the efficient maintenance and operation of coal cleaning plants.

J. Griffen—I would like to say something for the equipment manufacturers and designers of coal cleaning plants. I would like to compliment Mr. Hunter very much on his paper and I am delighted with the emphasis he placed on planning and on the fact that the matter ultimately depended upon management's willingness to spend the money for the plant and equipment required. That is where the responsibility resides largely because the builders must conform to the wishes of the purchaser and his ideas as to the amount of money he can spend.

Some years ago, we were entrusted with the contract to design and erect a coal preparation plant for a steel company which was accustomed to 24-hr operation each day in the year and knew that it cost money to build a plant to be operated on such a schedule with a minimum of shutdown time for repairs. This plant was projected to operate 20 hr out of each 24-hr day each day of the year. The plant we designed to meet this condition cost more money per ton of input than any bituminous coal cleaning plant previously built. During one stretch of 8 months, this plant was operated 22 hr out of each 24-hr day every day in the 8-month period. Such continuous operation was possible only because the steel company had spent considerably more money on their coal washer than was usual practice.

T. W. Guy—It was probably money well spent.

J. Griffen—That year it handled over five million tons of coal.

B. M. Bird*—Before I worked for Jeffrey, one coal company asked me to help with the design of a coal-cleaning plant. They said, "We want 250 tons an hour." I said, "Gentlemen, how much are you going to run through the plant?"

After some discussion they decided that the plant should be good for 300 tons an hour. I said, "What can you do to get production up to 350?" If they spent a million dollars underground they thought it might be possible to do so.

When I wrote up the specifications I wrote them for 400. Well, that plant did not run one week less than 400 tons an hour.

Now I am telling you this story with one motive. More often than not, undercapacity is faulty appraisal of the amount of tonnage that has to go through the plant. If you run the tonnage up 50 pct over what the plant was designed for, you will have a high-maintenance plant. I urge all of you who have anything to do with coal washeries to get the tonnage right. Maybe, after you think you have it right, you should add 50 pct, then you will be close to what you will probably require.

T. W. Guy—A factor of safety should be applied when setting capacity. Most plants that I have had occasion to know about have handled a great deal more than the design capacity. I think Pittsburgh Champion was designed for about 500 tons and they have run as much as 900 to 1,000 tons an hour through that.

R. E. Zimmerman*—Mr. Hunter's paper touches on a subject of increasing importance and of which too little has been said by preparation men in general. We may discuss frequently and learnedly concerning the theory of coal washing and the superiority of some pet processing equipment, but to the people who have to operate a cleaning plant day by day, and often 24 hr a day, the maintenance of the equipment is frequently of more vital importance than the particular kind of process used.

As cleaning plants are becoming more and more complex, of larger capacity, and frequently operating on a three shift basis processing coal from completely mechanized mines, it is paramount that there should be a minimum of mechanical delays. Many plants have little or no storage facilities between the mine and plant and even a few minutes delay may seriously affect production within the mine and prove very costly to management.

Maintenance begins with the design of the plant. Proper space should be allowed for access to machinery and equipment. Where practical, a plant should contain parallel flow of duplicate units so that when a breakdown does occur the plant can at least operate one circuit at reduced tonnage. This has value also when coal is slow in coming to the plant.

Equipment should be designed to take care of ample loads, as it appears to be almost universal that a plant is operated at higher tonnages than originally calculated. This is especially true in the sections dealing with the processing and handling of the fine sizes.

Serious consideration should be given to the use of alloy steels of both corrosion and abrasion resistance types. Rubber lined piping and chutes should be considered where practical.

Mr. Hunter brings out the importance of preventive maintenance. This is of great value not only in eliminating "delay time" but also in reducing the amount of major repair jobs necessary over weekends where one has to pay mechanical help time and one-half or double time.

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* Jeffrey Manufacturing Company.

* Hanna Coal Company.

It is always desirable, where possible, to use similar equipment, motors, and so on, throughout the plant in order to provide interchangeability and parts in common. It is frequently of value to use the "unit replacement" principle of repair, where a complete unit is installed rather than tearing the old one down in place. This will cut down time and provide for more leisurely and thorough repair of the equipment involved.

I agree most heartily with Mr. Hunter that the maintenance crew is no place to dump inefficient personnel. The wide variety of work involved, the need for skill and speed, the necessity for improvising and, at times, creative talent, require high caliber type of men above the run of the mill.

The maintenance foreman should be carefully chosen not only for his mechanical knowledge and steadiness of character but his ability to lay out and schedule his repair work in advance of actual need. He can make or lose the company a great deal of money in a very short time. Serious consideration should be given to the hiring of maintenance foremen who have a higher educational level than in the past, particularly in the large preparation plants where modern machinery needs technical supervision to produce the most efficient results.

Maintenance men proverbally dislike to keep records and if they do they are apt to be incomplete, or worse yet, never utilized. Here, again, the right type of maintenance foreman will make intelligent use of this tool and be able to save the company considerable money in knowing the rate of turnover of parts and equipment, anticipating machinery life and being able to prove to himself and his superiors that a certain piece of equipment actually has a high or low, as the case may be, maintenance cost. The writer, for instance, has had instituted at all Hanna plants a maintenance cost system which not only gives management comparative maintenance costs between plants but provides maintenance cost information on practically every type of equipment or process used by the company in its preparation plants.

R. M. Hunter (author's reply)—I am gratified that, in the discussion of this paper, the sentiments expressed are in agreement. If this large area of agreement can be converted into action, I believe that definite improvements in costs and product can be achieved.

I would like to re-emphasize my conviction that successful maintenance must start with design, and that all efforts possible should be made to assure adequate capacity and provision for maintenance. In this connection management has a governing voice.

I agree with Mr. Zimmerman's suggestion as to higher qualifications for maintenance supervisors. However, attracting and holding first class men for this work constitute a problem that has received too little thought. It is my feeling that this phase of the problem presents a very real challenge; especially in the present labor market.

Coal Dock Operations of the North Western-Hanna Fuel Company at the Head of the Lakes*

By J. T. Crawford,† Member AIME

(White Sulphur Springs Meeting, November 1948)

Although nearly 10 pct of the total tonnage of coal produced annually within the United States is handled by bulk freighters on the Great Lakes, very little of the detail connected with it has been published other than occasional newspaper stories and publication of tonnage statistics. Of the total tonnage floated on the Lakes each year some 10,000,000 is stored and distributed from the port of Duluth-Superior, at the western end of Lake Superior commonly known as the Head of the Lakes. This port has the largest single area concentration of coal docks in the world. Since this area contains the largest ore docks, the largest movable material handling bridge, the largest and highest grain elevator and the largest coal briquetting plant in the world, it is entirely fitting and proper that here also should be located the largest coal dock and what we believe to be the world's largest clam shell.

Of the sixteen coal docks operated by ten companies, five are owned and operated by the North Western-Hanna Fuel Co. which has two docks on the Superior, Wis., waterfront and three docks in Duluth, Minn. It is with these five docks that we are primarily concerned.

General History

In the summer of 1871 a small sailing vessel entered the harbor of Duluth-Superior with the first commercial coal cargo. All the coal brought up that first year did not amount to more than 3000 tons. During the year 1877 the first dock equipped for handling coal was built in Duluth. Coal receipts increased to 52,785 tons in 1879 the first year for which an official record was kept. Since then the volume of water-borne coal to the Head of the Lakes steadily increased to a maximum of 12,688,321 tons in the year 1923. This tonnage was nearly equalled in the year 1927 and the next highest tonnage recent year was in 1946 when 10,105,703 tons were unloaded. The average annual bring-up over a ten year period 1938 to 1947 was 8,605,231 tons. Approximately 30 pct of the coal unloaded at the Head of the Lakes is handled over the docks of the North Western-Hanna Fuel Co. Competition of other fuels coupled with expansion of coal fields in the mid-west have held coal receipts for Duluth-Superior at a relatively constant figure during the last eight years although the total tonnage of coal floated on the Great Lakes has more than doubled in the past 25 years.

From the shovel and wheelbarrow method of unloading early cargoes to the horsepowered windlass derrick with a wooden tub was but a short step. The first movable coal handling, steam operated,

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steel bridges were constructed on Rices Point, Duluth, during 1888 and the first clam shell bucket was used on a Superior dock about 1896.

As the tonnage increased rapidly between the years 1890 to 1910, the early dock operators' ingenuity was sorely taxed keeping in step with mechanical progress. Stationary unloading towers of varied types powered by mule team or steam and, in at least one instance, by a steam driven line shaft over 1800 ft long, were used in conjunction with hand loading of the steel tubs that replaced the earlier wooden ones. These were closely followed by the addition of cable car systems for transportation of the coal from the unloading towers to the dock storage areas. The stationary wooden towers were replaced by movable steel towers, either steam or electrically driven, and these were in turn replaced by movable steel bridge spans equipped with rope or man trolleys and the clam shells that are in use today. The early operation of unloading boats was very crude in most instances. At that time stevedores shoveled the coal first into wooden tubs and later into steel self-righting tubs. These were then hoisted first with horses and later with steam hoists, carried back over the storage area and dumped into stock piles, first with wheelbarrows and later with push cars and trolleys on bridges. This method lasted until about 1900. Many of the docks prepared bituminous coal for shipment by screening the coal over stationary inclined screens similar to the type used by plaster tenders in screening plaster sand. This screened coal was then shoveled into wheelbarrows and dumped into box cars. This operation was used as late as 1910.

About 1905 coal dock activity and expansion started at an increased pace. The clam shell bucket for handling coal had been perfected. Most of the existing docks were old and obsolete and the market for Lake coal was expanding. The coal companies began rebuilding their docks and enlarging and equipping them with more modern coal handling machinery. Many new docks were constructed. This activity and expansion continued until about 1922. Since then only three coal handling bridges have been built; one in 1926 and two in 1929. Since 1930 this trend has been reversed as some bridges have been removed on docks where reduction of tonnage or addition of auxiliary facilities made all of the original structures no longer necessary. In three instances docks formerly handling coal have been converted and are handling either grain screenings or limestone.

At the present time it is possible to see equipment erected as early as 1905 almost side by side with modern structures erected in 1929 with all of the variations that occurred between those dates. During the era of intensified construction and over-expansion between 1905 and 1915 many tremendous sheds were erected on docks primarily for the covered storage of anthracite coal. These structures covered a floor area as much as 240,000 sq ft. With the decline in anthracite tonnage after 1930 many of these were torn down; of the four remaining only one is used for anthracite, the others covering grain screenings or domestic stoker coal.

It is noteworthy that few, if any, fundamental changes were made in coal bridge structural design between 1912 and 1930, most of the changes to the bridges themselves during that period having been confined to the method of propulsion and to maintenance detail. The early bridges were usually propelled by a single motor coupled to all the legs of a bridge by a long and exceedingly jointed lineshaft system comprised of literally hundreds of universal joints and bevel gears, while the later built bridges are moved by individual electrically interlocked drives on each bridge leg.

Practically all of the progress during later years has been directed toward increasing the capacity of existing structures and in improvements to or building of better screening plants and loading facilities. Prior to 1910, the bulk of the tonnage consisted of three sizes: Mine Run, 1¼-in. Lump and 0 × 1¼-in. Screenings. Between 1910 and 1930 occurred the development of the sizes known as Block, Lump, Egg, Stove, Stoker, 2-in. Screenings, 1¼-in. Screenings and ¾-in. Screenings with a slight but very definite trend during the past 15 years away from the Block and Lump sizes toward the Egg, Stove and Stoker sizes. During the 1910–1930 period all screening was accomplished by shaker and trommel screens. From 1930 the trend toward more screening has continued with particular emphasis on the stoker sizes. Whereas formerly a great deal of tonnage consisted of Mine Run and Lump sizes, the bulk of the tonnage tends to trend to the minus 3-in. sizes including both the Stoker grades and Steam Slack. This trend is almost entirely responsible for the present modern screening equipment now in use on the coal docks. Those docks which have not modernized their screening equipment have their smaller sizes prepared entirely at the mines and stock them on the docks. As a result, regardless of the type of equipment used, the preparation and sizes of coal loaded by all docks at the Head of the Lakes are reasonably comparable. The docks which have not modernized both their screening plants and handling equipment have been penalized by higher operating costs as well. Modernization of screening equipment in most cases has not included the replacement of the original shaker screens as this equipment is not only satisfactory but desirable for the preparation of the larger sizes of Lump and Egg or any +2½-in. size. Trommel screens which were formerly used for preparation of smaller sizes have been entirely replaced by vibrating screens of various types. Choice of vibrating screens on most docks has largely depended upon available space in existing structures.

SUBSTRUCTURES

All five of the North Western-Hanna Fuel Co. docks at Duluth-Superior are constructed with sand and rockfilled timber cribs for dock facing next to the boat slip and three have a concrete dock front construction above low water datum. This construction was used at a time when hemlock timber could be purchased for $18.00 to $20.00 per 1000 and labor rates ranged from 20 to 40¢ per hr. The cost of duplicating this crib construction today would be prohibitive. The accepted type of dock front construction used today is a steel sheet pile dock front with timber wales for dock and vessel protection. It is the type of dock front used recently in building coal docks at other lake ports. At some of the docks the original timber cribs were only 17 to 18 ft deep below low water datum as that was apparently sufficient in 1900–1910 for the vessels then operating on the Lakes. Vessel slips today must be dredged to a depth of 22 ft to accommodate the desirable large freighters of 10,000 to 18,000 ton carrying capacity. This condition requires careful dredging in order to leave a rising bank of sand along the edge of the slip for the cribbing to rest on. Should a vessel overwork its propellor while lying alongside, the undercurrents set up can easily suck hundreds of yards of sand fill from behind a shallow cribbing in a matter of minutes. A severe storm can also remove hundreds of yards of fill from underneath such construction particularly if the shoulder on which the crib sits is too narrow. Fortunately such occurrences are not frequent. Cribbing or sheet piling are usually prevented from overturning or bulging outward into the slip by tie rods spaced at regular intervals and anchored either to dead man anchors or to the piling used to support the bridge runway foundations.

The bridges at Docks No. 2 and 3 require three runway foundations because of the use of a gantry or portal type pier instead of an inverted pier for the bridge. Bridges at Docks No. 1, 4 and 5 are of the inverted pier construction type and hence require only two runway foundations. Without exception, runway foundations are all set on piling. The bridge runway rails at Dock No. 2 are set on a 1- × 12-in. steel plate fastened to the two 12 × 14 in. timbers supported by the piling. The other four docks have a concrete runway foundation supported by the piling. A 1- × 12-in. steel plate rests on the concrete and in turn supports the rail on which the bridges travel. The 1- × 12-in. plate was not included in the original construction but has been added in recent years to reduce maintenance on both foundation and rail. Virtually all of the original rail, section 10040 in 33-ft lengths, is in the process of being changed over to section 11228 in 66-ft lengths. Since with individual bridges the wheel load may vary from 50,000 to 96,000 lb per wheel it is not uncommon to find a "bead" of as much as ¼ in. on either side of the ball of the rail where it has been rolled out and over from the top surface of the rail because of the heavy wheel loading. Bridges supported on three runways travel on a single rail per runway; and bridges supported on two runways invariably travel on dual rails set on each runway at about 30-in. ga.

The filling of all five docks is mostly clean sand originally pumped in from the bay or river channel. Where this is the case, any water occurring on the docks readily drains down through the sand and full advantage of this is taken whenever possible. Dock flooring on top of the sand filling was formerly 2-in. planking which is rapidly being replaced by 6-in. non-reinforced concrete. Wooden flooring ordinarily was given a life use of 7 years but this was largely determined by how long and when the wooden flooring was exposed to the sun or when a section of dock floor could be vacated for a sufficiently long period to relay an area. At the present writing 6-in. non-reinforced concrete can be laid for 2 to 3¢ per sq ft more than 2-in. maple hearts and concrete floor experience has been good over a 25 yr period. While clean-up of a concrete floor is harder on clam shell bits, large areas can be easily cleaned up and scattered ridges repiled with bull dozers or preferably rubber tired industrial tractors fitted with various makes of scoop buckets. Clean-up with this type of equipment is simple on concrete flooring but on wooden flooring the process is slower as more care must be taken not to start pick-up of the wooden floor planks. Bunching loose coal with this smaller equipment on concrete floors relieves the big bridges of a great deal of slow, time consuming work.

Bridge Structures

The two bridges on Dock No. 1 (Fig 1) and the one bridge each on Docks No. 4 and 5 (Fig 4 and 5) are of the inverted pier type while the three bridges on Dock No. 2 (Fig 2) and the two bridges on Dock No. 3 (Fig 3) are of the gantry or portal pier construction. (See also Fig 6, 7, 8, 9 and 10). While the gantry pier construction was believed to be outmoded between 1915 and 1935 it is interesting to note that some ore bridges built recently have the pier end made up of gantry construction on which is superimposed an inverted pier construction. All nine bridges are of the skew type with a skew limit of approximately 5 pct. All have 100 pct drive on all truck wheels. Bridges at Docks No. 1, 4 and 5 have fixed cantilevers and all bridges at all five docks, of course, have lift booms at the slip or unloading end. The bridges at Docks No. 2 and 3 have their main screening plants within or attached to the gantry pier ends which are at the unloading or slip ends of the bridges and those at Docks No. 1, 4 and 5 have their screening plants located within or attached to the inverted piers which also support the fixed cantilevers on the ends away from the vessel slips. Docks No. 1 and 2 have loading hoppers and box car loading

TABLE 1—*Comparative Bridge Data*

Make	Dock No. 1		Dock No. 2			Dock No. 3		Dock No. 4	Dock No. 5
	Bridge No. 1	Bridge No. 2	Bridge No. 1	Bridge No. 2	Bridge No. 3	Bridge No. 1	Bridge No. 2	Bridge No. 1	Bridge No. 1
	Industrial Brown Hoist		Heyl and Patterson			Industrial Brown Hoist		Brown Hoist	Brown Hoist
Year Erected	1912	1912	1912	1909	1910	1910	1910	1920	1921
Span length in ft	342	342	363	363	363	242	242	342	320
Overall length-ft	589	589	479	455	455	331	331	571	549
Apron length-ft	60	60	70	70	70	56	56	81	75
Cantilever length-ft	125	125	24	none	none	none	none	148	146
Runway length-ft	2,857	2,857	1,963	1,963	1,963	1,572	1,572	1,556	2,100
Height trolley rail-ft	71	71	68	68	68	64	64	75	75
Traverse Speed fpm	59	59	70	70	70	61	61	87	90
Trolley Speed fpm	1,000	680	1,200	1,200	1,200	900	900	1,000	730
Hoist Speed fpm	287	247	300	300	300	225	225	200	212
Turntable on trolley	Yes	Yes	No	No	No	Yes	Yes	Yes	Yes
Height of Coal	50	50	42	42	42	40	40	49	49
Capacity per hr (tons)	600	400	400	400	400	400	400	500	600
Skew Limit (Maximum Permitted)	36 ft	36 ft	40 ft	40 ft	40 ft	26 ft	26 ft	35 ft	32 ft
Skew (either way) (setting)	12 ft	12 ft	10 ft	10 ft	10 ft	10 ft	10 ft	15 ft	15 ft
Screening Plant Location	Pier	None	Pier	None	None	Pier	None	Pier	Pier
Kind of Screens	Shaker		Shaker and Vibrator			Vibrator		Shaker and Vibrator	Shaker
Screen area (Shaker screens)	168 sq ft		169 sq ft					86 sq ft	90 sq ft
Elevator capacity tons per hr	540		300					336	425
Elevator speed fpm	186		207					177	125
Power	550 DC	550 DC	440 volt 25 cycle			440 volt 25 cycle		550 DC	550 DC
Power Demand (entire dock)	432 KW		564 KW (2 bridges only)			337 KW		484 KW	426 KW
Motor (Bridge moving)	150 HP	150 HP	2 at 40 HP and 1 at 75 HP each bridge			112 HP	112 HP	210 HP	225 HP
Motor-Holding (Shell) Line	187.5 HP	230 HP	150 HP	150 HP	150 HP	225 HP	225 HP	210 HP	225 HP
Motor-Closing (Hoist) Line	187.5 HP	same motor	150 HP	150 HP	150 HP	same motor	same motor	210 HP	225 HP
Motors—Rack	2 at 90 HP	2 at 50 HP	2 at 75 HP	2 at 75 HP	2 at 75 HP	2 at 75 HP	2 at 75 HP	2 at 100 HP	2 at 110 HP
Orig. Permitted span load (tons)	50	50	44	44	44	48	48	75	76
Pres. Permitted span load (tons)	56	50	44	44	44	48	48	81	84
Weight of Bridge (tons)	959	878	540	490	490	388	388	1,191	1,121
Original weight of trolley	37	37	28	28	28	36	36	52	52
Present weight of trolley	34	35	28	28	28	33	33	52	52
Orig. wt. clam and load (lb)	25,800	25,800	31,000	31,000	31,000	24,400	24,400	46,000	48,000
Pres. wt. clam and load (lb)	52,000	31,000	31,000	31,000	31,000	31,000	31,000	59,000	64,000
Orig. clam capacity (tons)	6	6	6	6	6	6	6	10	12
Present clam capacity	14	8	8	8	8	8	8	16	17

equipment in the shear legs but Docks No. 3, 4 and 5 do no loading whatsoever through the shear legs. Dock No. 3 bridges have short shear legs traveling on an elevated A frame trestle as the single span bridges on this dock were once the first spans of three-span bridges.

FIG 1

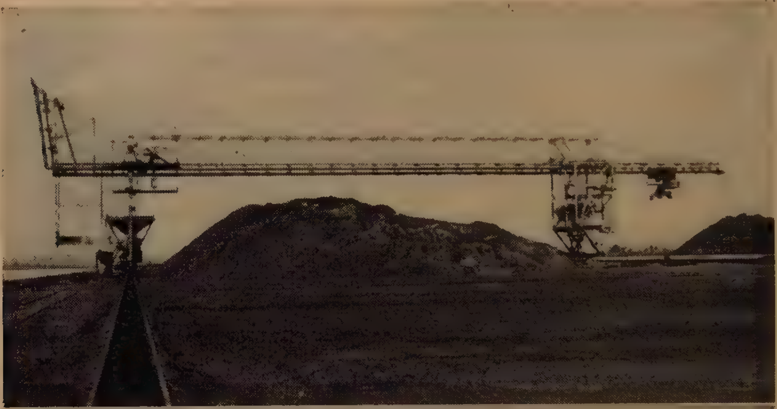


FIG 2

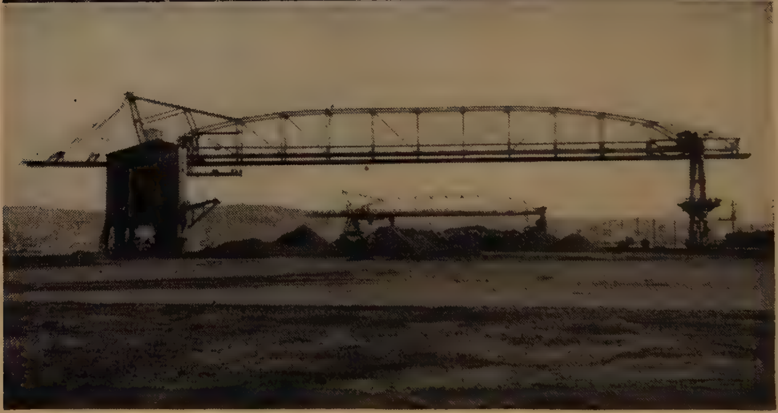


FIG 1—BRIDGE NO. 2 AT DOCK NO. 1.
FIG 2—BRIDGES AT DOCK NO. 2.

The pier leg is the stability member of each bridge and if of the inverted type supports the truss structure of the span at four points on the top of the inverted pier. These points of support consist of flat plates, generally bronze plates sliding on steel, but often of steel on steel and provide a surface for the superstructure to slide on the pier leg when the bridge is skewed. On the gantry or portal pier type the span is supported by two sets of steel plates separated by tapered steel rollers. In either case a king pin, centered between the two support points of the gantry or portal type pier or between the 4 points of the inverted type pier at the center of rotation, serves to keep the truss and pier leg sliding plates in alignment.

The shear legs consist of a single plane construction from the top cross members of which the shear end of the span may be suspended by two eye bars. A king pin centered between the two eye bars keeps the shear end of the span centered in the shear. Since the holes in the span members admitting the king pin are slotted longitudinally with the center line of the bridge,

FIG 3

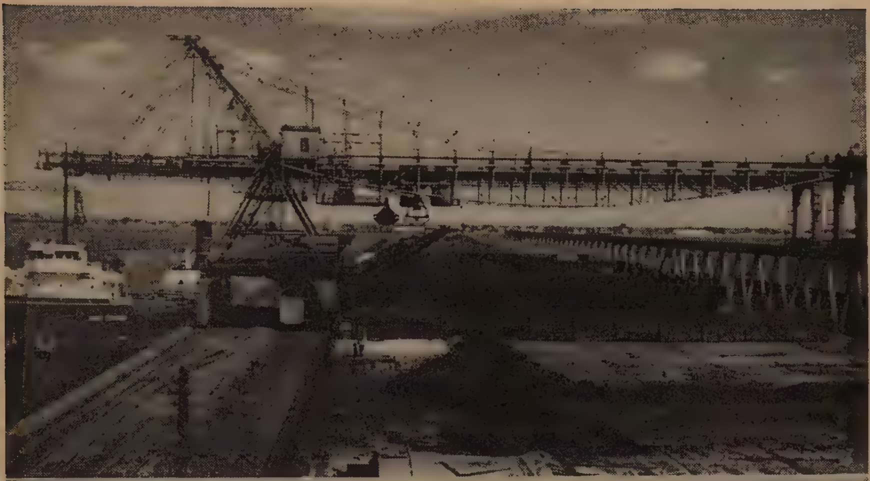


FIG 4

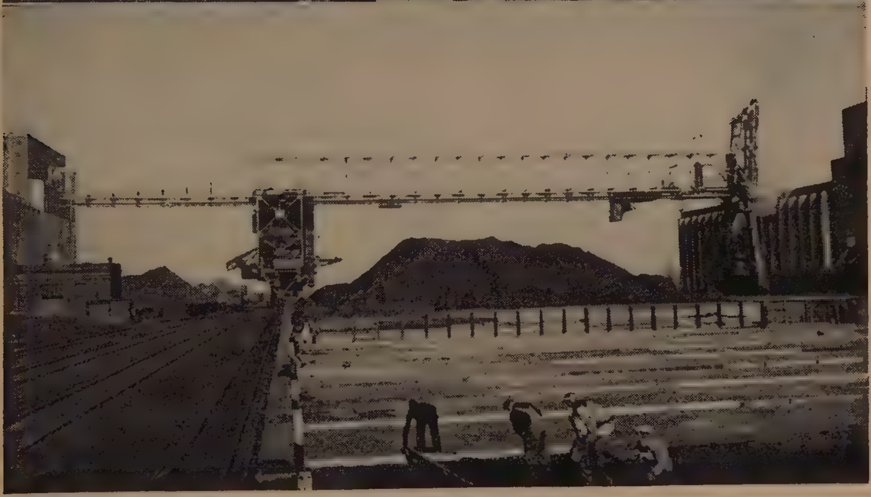


FIG 3—BRIDGES AT DOCK NO. 3.
FIG 4—BRIDGE AT DOCK NO. 4.

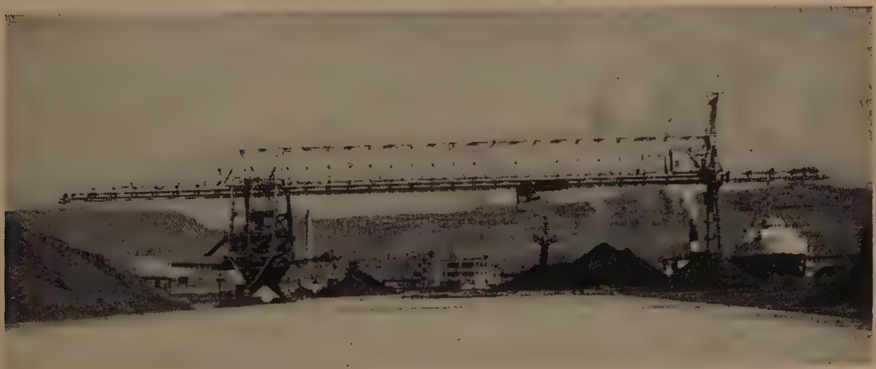


FIG 5—BRIDGE AT DOCK NO. 5.

the bridge span may expand and contract without rocking the bridge on the pier legs. This feature also is extremely useful when the shear and pier runways get out of gauge as not infrequently happens; and it also permits inclination of the shear leg when the bridge skews. The more frequent and better type of construction has the top of the shear leg terminating at a point on the centerline of the bridge structure. In this case the superstructure is supported at this end on a single ball and socket joint generally of steel to steel contact but frequently having a bronze bushed socket in which a hemispherical steel casting fits. Lubrication may or may not be provided at such a point. This type of construction provides any necessary play for the horizontal movement and the inclination of the shear leg if the bridge skews. A king pin provides against tipping or teetering of the span transversally.

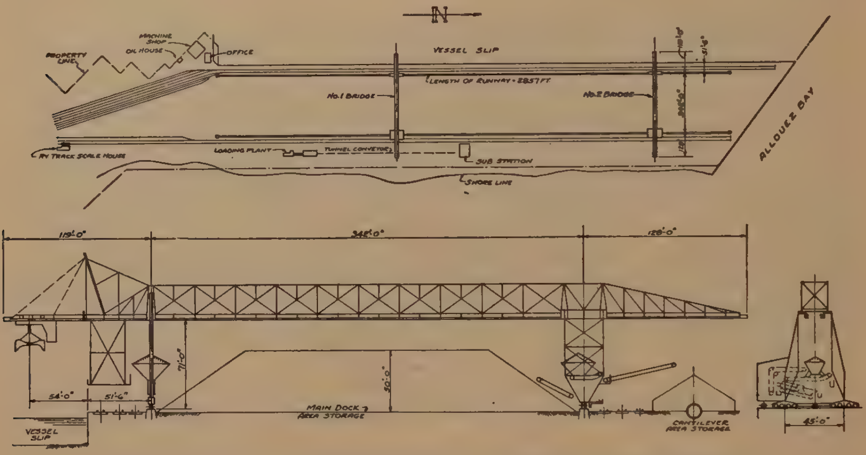


FIG 6—DOCK NO. 1—(ABOVE) AREA PLAN. (BELOW) CROSS-SECTION.

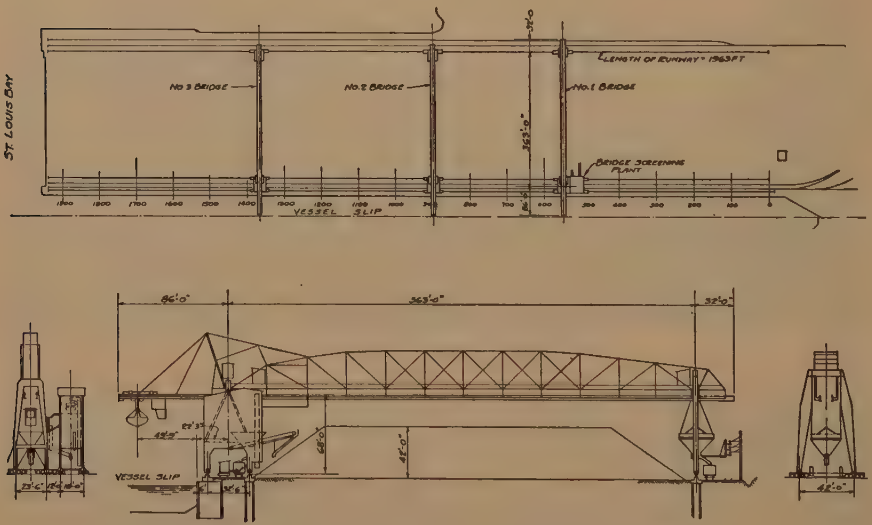


FIG 7—DOCK NO. 2 (ABOVE) AREA PLAN. (BELOW) CROSS-SECTION.

The Dock No. 2 structures are stiff connected bridges while Docks No. 1, 4 and 5 structures are link and pin connected. The Dock No. 3 bridges, which were formerly 3 span structures, have been shortened in length by dropping the middle and inshore end spans. They are of the catenary truss type which, while considerably more flexible and "floppy," are quite as satisfactory as the other type except that obviously the future maintenance on such structures will far exceed that of the other bridges because of so many points of wear in the pin connections.

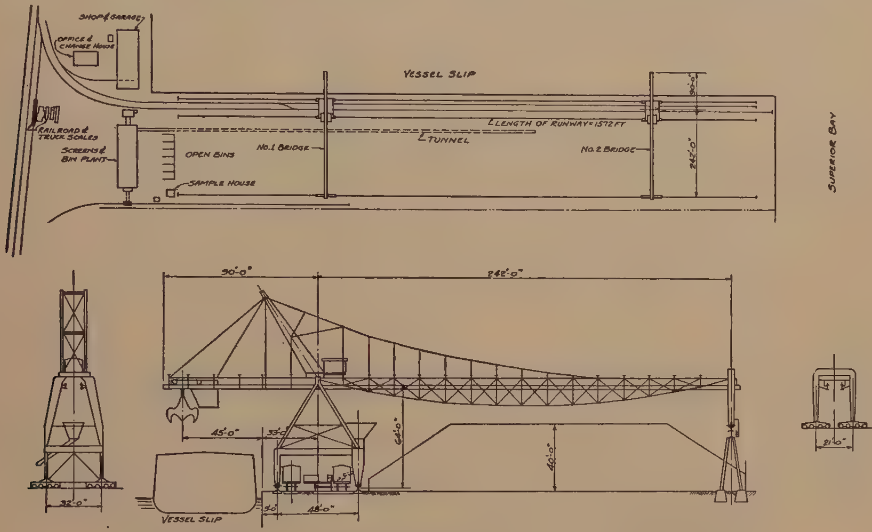


FIG 8—DOCK NO. 3—(ABOVE) AREA PLAN. (BELOW) CROSS-SECTION.

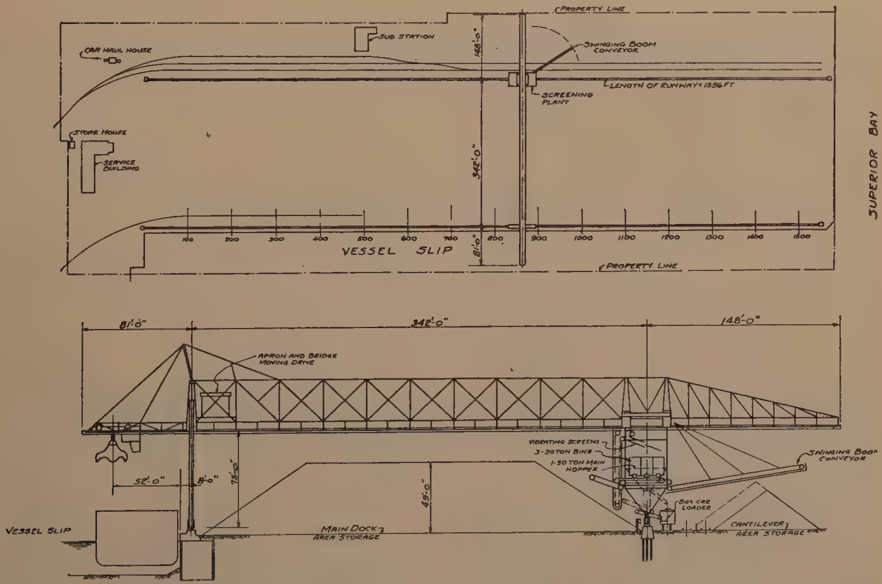


FIG 9—DOCK NO. 4—(ABOVE) AREA PLAN. (BELOW) CROSS-SECTION.

With the exception of Dock No. 2 all the bridges are driven by a single motor on each bridge geared to all wheels (100 pct drive) by a complicated and many jointed series of line shafting working through many universal joints and bevel gears. This sort of arrangement is burdensome to maintain and lubricate. The expense of changing over to individual drives on each leg in accordance with more desirable practice is prohibitive at the present time. Dock No. 2 bridges are driven from three points. A motor drives the truck wheels on each leg of the shear legs and the pier end is driven by one motor operating through lineshafting and bevel gears to each of the four corners of the gantry pier.

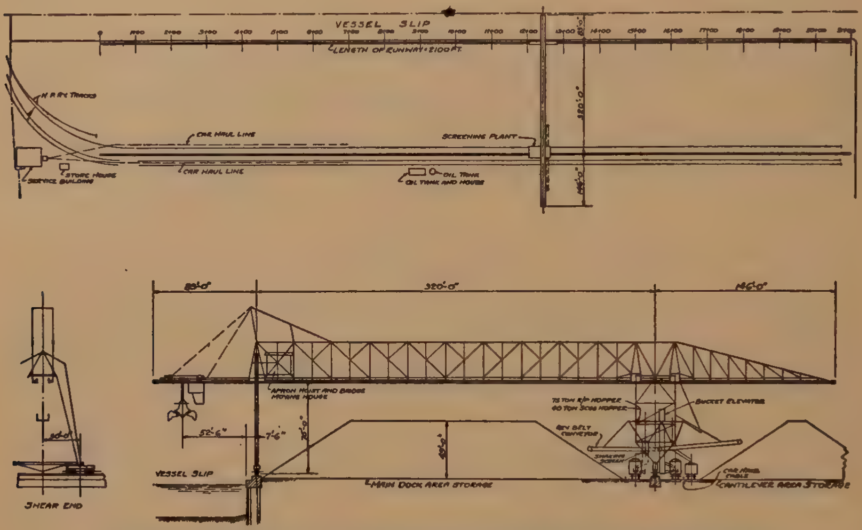
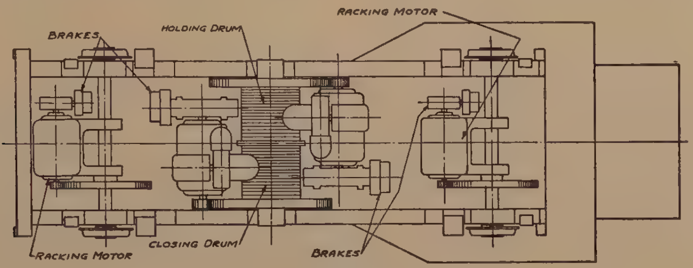


Fig 10—Dock No. 5—(above) area plan. (Below) cross-section.

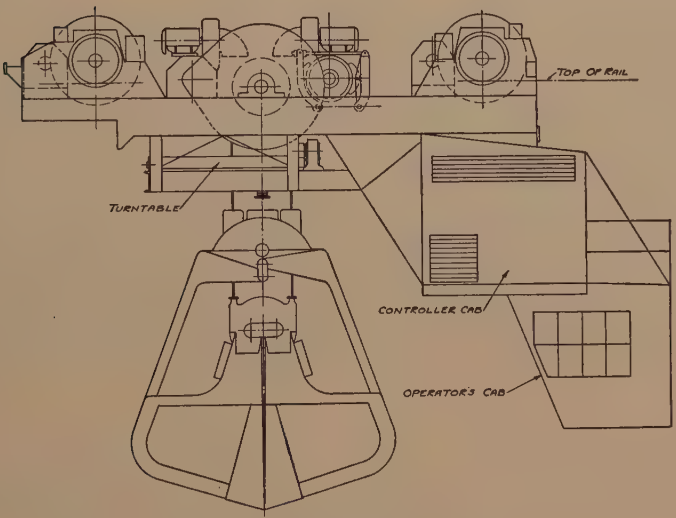
Trolley track construction is far from satisfactory. If new bridges were to be built it would be preferable to anchor the track rail to the track beams by using tie plate sections bolted or welded to the span and any one of several types of spring held clips to clamp the rails to the tie plates following the same procedure as used by railroads where endless rail is used. In the original construction of the bridges the trolley track rails were riveted to the track beams but this is not desirable as it is impossible to keep rivets tight under the stresses involved. This construction has been replaced on all of our bridges. In one instance the rivets have been replaced with ordinary machine bolts equipped with self locking thread which has been reasonably satisfactory; another bridge had the rivets replaced with tight driven ribbed bolts with self locking threads and nuts which are no better than the machine bolts with a self locking thread. A third method of stitch welding the rail to the track beams has been used on seven of our bridges with what might be called mediocre results since there are usually several welds broken and rewelding is required most of the time. In all instances the best results have been obtained where the trolley rail is butt welded to form a continuous rail.

Trolleys

All nine bridges on the five docks are of the man trolley type (Fig 11). Since all clam buckets are of the two-line type each trolley carries two rope drums either on the same shaft and driven by one motor through clutches and friction bands as is the case with both trolleys at Dock No. 3 and one trolley at Dock No. 1, or each drum (both the hoisting line and the holding line) is driven by its own motor.



PLAN VIEW



SIDE ELEVATION

FIG 11—TYPICAL MAN TROLLEY LAYOUT.

Travel motors on one trolley at Dock No. 1 are supported in cradles attached to the axles of the trolley via roller bearings and permit the continuous engagement of the driving pinion and axle gears at fixed center distances regardless of the spring action of the trolley proper. This attachment of the motor cradles integral with the axle rather than with the trolley itself provides proper engagement of the teeth of the motor pinions and axle gears at all times. On this particular trolley it permits the use of continuous herringbone gears thus providing a most satisfactory and nearly noiseless trolley travel since the trolley suspension is of the spring type permitting considerable vertical movement between the trolley frame and the axles. Unfortunately all the other trolleys have the motor bases situated directly on and in the trolley frames so that the motors move with the trolley frame. While the varying center distances between the motor shaft and axle shaft are short and not too disturbing to spur gearing, this does not lend itself to herringbone or spiral gear teeth. Change over to the other type of motor mounting is not economically feasible since redesign of the entire trolley would be required. The one trolley at Dock No. 1 previously referred to is a newer type trolley constructed of light weight materials in 1935. Trolley wheels are all of rolled steel, rim toughened. With the exception of the trolleys on the Dock No. 2 bridges, all are equipped with turntables of one kind or another permitting rotation of the bucket through a 90° or more arc to enable better clean up of a vessel during the unloading operation after the clam has been lowered through the hatch opening. On the original trolleys at Docks No. 1 and 3 this is accomplished by rotating a heavy turntable which carries the hoist motor and both the closing and holding line drums. On one trolley at Dock No. 1 and the trolleys at Docks No. 4 and 5 the two rope ends are anchored to a ring turntable rotated by a worm gear. Rotation of the rope end ring provides rotation of the clam bucket.

In the three older trolleys at Docks No. 1 and 3 the operators and the control panels share the same cab. In the other trolleys the operator's cab is located beneath the control room and hence is better situated for both visibility and safety of the operator.

The trolleys at Docks No. 1, 4 and 5 are equipped for full dynamic braking since these docks are operated on 550 volt dc current. The trolleys at Dock No. 2 are powered with 440 volt ac (25 cycle) and since the control contactors are air operated the trolleys are equipped with air brakes. At Dock No. 3, also powered by 440 volt, 25 cycle ac, all brakes on the trolleys are hand operated as available space does not provide for electric or air brakes. The material result of this is that at Dock No. 3 all braking of both hoist and rack is done by plugging the motors which gives rapid operation. Since the motors and controls recently installed are designed for this, no harm is done. Normal operation at the other docks includes much plugging of the motors since this results in faster operation and less operator fatigue. Only at Dock No. 2 are the brakes used consistently.

Some modernization has been accomplished on the trolleys themselves. The present trolley on No. 1 bridge at Dock No. 1 replaced the entire original trolley. Primarily it was designed and built to reduce trolley weight so that a larger capacity clam could be used without much increase on the bridge structure itself. The trolley is entirely modern, being built of aluminum and low alloy steel. It has fan cooled ribbon resistances, enclosed herringbone gearing, and all bearings of the anti-friction type. The net result, coupled with a slight reinforcing of the bridge structure itself and including a light weight clam shell, increased the effective capacity of the clam bucket from an original 6 tons to its present 14 ton capacity. On the No. 2 trolley at Dock No. 1 a recent replacement of the control panels plus replacement of the original cast grid resistors with edge wound resistances resulted in a net weight saving of 4125 lb. On this bridge a new light weight clam has increased the clam capacity from 6 tons to 8½ tons without increasing the gross weight of the loaded clam. The partial modernization of the trolley has not only increased the capacity but has effected a substantial saving in weight on the span which should be reflected in decreased maintenance of the bridge structure and is deemed to be a valuable saving in view of the age of these 1910 structures. Modernization of the Dock No. 3 trolleys and replacement of the clams with light weight buckets have increased the capacity of the buckets from 6 tons to 8½ tons and effected a saving in weight on the trolley alone of 7138 lb. This was accomplished by new control panels, edge wound resistances in place of the previous cast grid resistors and a change in the rack gear ratio which increased acceleration and reduced an unused top speed thereby permitting use of 75 hp motors in place of the original 112 hp. In this case also the net weight saved in the trolleys reduced the weight on the span since the increase in clam capacity resulted from the weight saved in the clam itself plus some minor reinforcing to the bridges. The natural question here would be to ask why the 7138 lb of weight saved could not be applied to a larger clam. This is not advisable for two reasons. The age of the bridges makes it expedient to reduce the weight on the spans; and the additional 3.5 tons, added to the closing and holding ropes, would measurably shorten the life of those cables since they are operating with a factor of safety of only 2.4. Increased diameter of the ropes would require new and larger drums which would necessitate entirely new trolleys which would not be warranted at present construction costs on spans of the type and age of those at Dock No. 3.

Of the nine trolleys only the light weight trolley at Dock No. 1 was originally constructed with roller bearings. All of the others have been changed from the original oil journals on the trolley axles to roller bearings; and other bearings are in process of being changed over.

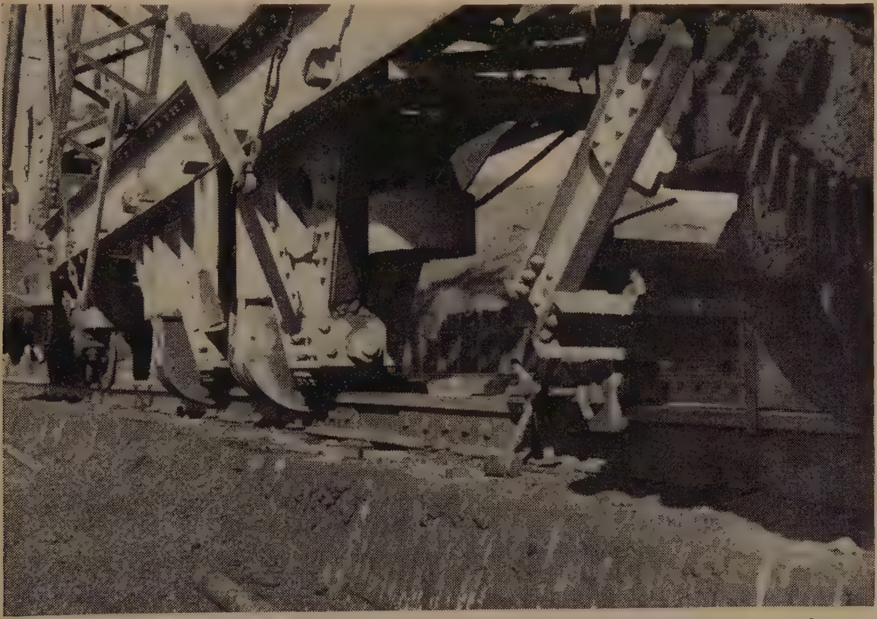


FIG 12—AUTOMATIC CLAMPS, STORM STRUT, AND WHEEL WEDGES IN PLACE WHEN BRIDGE IS IDLE OR TIED UP BECAUSE OF HIGH WINDS.

GENERAL BRIDGE SAFETY FEATURES

Not the least of the auxiliary items attached to parts of the bridge structures and trolleys for the purpose of protecting the equipment and operating personnel are such items as hoist limit devices, skew limit devices, trolley bumpers, anemometers and other wind velocity indicators, boom latches, rail clamps, storm struts, wheel wedges, and others.

Rail clamps (Fig 12) are of very simple construction in that primarily they consist of a very heavily constructed and powerful set of toothed jaws anchored to the sill of the pier and shear legs and so arranged as to clamp firmly to the ball of the bridge travel rail. The clamps are either spring or weight set and motor released. Clamps are installed to operate independently of the bridge operator in case of winds over 35 mph velocity—the electrical controls for this feature being activated either by anemometers or swing paddle wind gauges. In addition, the clamp controls are interlocked with the bridge travel controls so that the clamps set automatically when bridge travel stops and release prior to the start of it. In case of power failure or interruption of the bridge travel power by the anemometers in case of high winds the clamps set automatically. Adjustment of the clamps on each end of each bridge is tested periodically with full power on the bridge moving motors. Clamps should withstand this sort of test without sliding or loosening and thus prevent the bridge from moving.

During periods of non-operation of the bridges, either during a normal shutdown or a stoppage of operation due to high winds, wooden or steel wedges are placed at each wheel to deny rolling movement to the bridge travel wheels; and storm struts (strong steel section bars hinged to the pier and shear sill ends) are clamped or bolted by hand to the bridge rail or to its foundation and serve as additional and supplementary aids to the main automatic clamps.

Hoist, skew and bridge travel are fitted with limit switches to prevent in each case: hoisting too high; the bridge from skewing past the danger point with the travel power on; and the bridge moving into the bridge rail bumpers with the travel power on. It is taken into consideration with all of these devices that the operator may be engaged in some other activity, become momentarily absent minded, or incapacitated for some reason while the bridge, trolley or hoisting drums are in motion. Therefore "dead man" or self centering controls are installed wherever possible.

Two types of trolley bumpers are in use: the swinging beam or squeeze clamp type which bears on the trolley wheel, and the spring box type which absorbs the thrust of the trolley frame.

Anemometers, both of the Burton rotating cup type, recording and non-recording, or of the Stenbol swing paddle type, are provided for each bridge and in each case make contact with the necessary controls to cut off all bridge moving or travel power and to set the bridge clamps at wind velocities in excess of 35 mph.

Boom latches are provided on each boom mast so that when the boom apron cantilever sections are raised, as they all are during the navigation season except while actually unloading vessels, the booms are safely held in the raised position without any strain on the boom raising and lowering cables.

BUCKETS

Clam shell design and the materials used in the construction of them changed but little between 1910 and 1935. Since that time much has been accomplished both in increase in maximum capacity and a decrease in dead weight to live weight ratio. Whereas at one time a 1.25:1 ratio was acceptable it is now commonplace design and construction to have a clam weight to pay load ratio of 1:1.25. Clams have been operated where the ratio was as high as 1:1.34 but with the extremely hard service given these buckets the maintenance was judged to be excessive and it was found necessary to add sufficient reinforcement to provide for longer periods of uninterrupted service. Addition of the necessary reinforcing provided a satisfactory clam but reduced the weight ratio to 1:1.25. By the use of such light weight clams which reversed the dead weight to pay weight ratio the capacity of the bridges in unloading or reloading of unscreened coal has been increased considerably without increasing the weight on the bridge spans or requiring larger hp motors. Where it was deemed advisable to try to increase the use life of the bridges themselves the substitution of lighter weight buckets of the same capacity as those replaced gave a net reduction in the

TABLE 2—*Bridge Clam Data*

Dock	Kind	Bear-ings	Material	Rated Capacity		Empty Weight (lb)		Contents Weight (lb)			Overall Width		Height Open		Length Open		Rope Size
				Cu Ft	Tons	Original	Present	Screenings	Lump	Stoker	Ft	In	Ft	In.	Ft	In.	In.
								36,120									
5	Digging	Roller	Low alloy Steel	560	16	28,020	28,020	35,750	20,130	32,900	7	6	19	0	22	0	1⅛
								32,000									
4	Digging	Roller	Low alloy Steel	520	15	27,200	27,200	32,340			7	6	18	6	21	3	1⅛
1	Digging	Sleeve	High alloy Aluminum	470	12½	22,200	24,080	29,000	16,100		7	6					1
									20,530								
5	Digging	Sleeve	Mild Steel	480	12	28,100	28,100	26,960	24,100	24,625	6	6	15	9	22	0	1⅛
									17,470								
4	Digging	Sleeve	Mild Steel	400	10	26,900		21,800	19,200		6	6	17	8	22	0	1⅛
3	Digging	Sleeve	High alloy Aluminum	320	8½	12,660	14,260	16,600			7	3	14	3	15	6	⅞
1	Digging	Roller	Low alloy Aluminum	300	8½	14,000		17,160			7	3	14	4	16	1	⅞
2	Digging	Roller	Low alloy Aluminum	300	8½	14,000		17,160			7	3	14	4	16	1	⅞
3	Digging	Roller	Low alloy Aluminum	300	8½	14,000		17,160			7	3	14	4	16	1	1
2	Digging	Sleeve	Mild Steel	300	8	16,390	20,100	17,100	14,000	14,520	5	9	13	2	16	3	1
								17,300									
2	Digging	Sleeve	Mild Steel	270	7	16,760	19,100	16,600	Anth.								
								16,400	19,300		6	1	13	10	16	10	1
1	Digging	Sleeve	Mild Steel	250	6	17,000	18,000	14,600			6	6	13	2	17	8	⅞
3	Digging	Sleeve	Mild Steel	230	6	17,000		12,000			6	6	13	0	17	9	⅞
5	Clean-up	Sleeve	Mild Steel		6	29,000		12,000			12	1	8	5	25	8	1⅛
4	Clean-up	Sleeve	Mild Steel		6	29,000		12,000			12	1					1⅛
2	Clean-up	Sleeve	Mild Steel		4	22,000	23,110	8,000			9	6	8	8	24	2	1

Note: Contents Weight in each case is average of 10 or more full clams.

live load on the span. With the clams now operating the maintenance is but little more with the light weight buckets than with the original heavier ones. All of them have had a few "bugs" which have been patiently removed. Particularly at Docks No. 1 and 3, where clean-up clams are not used, the digging clams not only handle all the coal off the dock including the "clean up" on the dock floor but also handle all coal out of vessels including the clean up off the steel bottom of the lake freighters. At Docks No. 2, 4 and 5 clean-up clams are used for finishing up unloading of vessels and also during the spring of the year in cleaning up the dock floors thus relieving the digging clams of a great deal of shock and jolting. (Fig 13). The subject of clean-up clams is frequently controversial and it is not proposed to expand the subject here. Suffice it to say that where trolley turntables are available and where auxiliary equipment such as tractors can be placed within the boats to feed the digging clams during the clean up period there seems now to be little justification for equipment such as clean-up clams which have such a limited use for other phases of the operation.

Fig 13

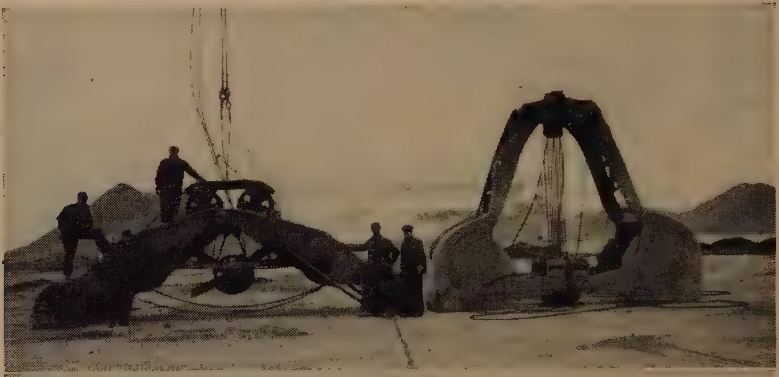


Fig 14

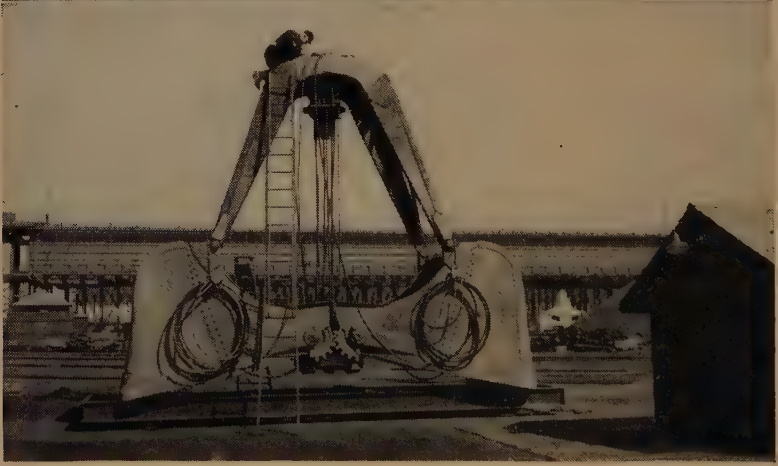


Fig 13—4-ton capacity clean up and 8-ton capacity digging clams, Dock No. 2.
Fig 14—22-yard clam at Dock No. 5.

As far as is known, the clams at Docks No. 4 and 5 (Fig 14) are the largest on the Great Lakes and probably the largest in the world. Larger clams could be built but there would be little use for them since radical changes in naval architecture would be necessary to accommodate buckets larger than the 22-yd clam now in use at Dock No. 5. Most of the clam buckets in use have been limited to 6 ft 6 in. in width to provide ready entry into the 8 ft wide hatches which were so prevalent in early vessels. Since the greatest percentage of bulk freighters plying the Lakes now have 9 ft hatches the use of 7 ft 3 in. and 7 ft 6 in. wide clams has caused us no concern and they have been used in case of necessity on 8 ft wide hatches although this requires accurate spotting of the bridge and a hoister with a very light touch. Clams with a width of 6 ft 6 in. having a capacity larger than about 12 tons of bituminous coal are in the writer's opinion not feasible since at about that capacity the ratio of opened length to free inside width becomes too great and the clams will not dig properly nor fill to capacity. One major asset of the 7 ft 3 in. and 7 ft 6 in. width clams as compared to the 6 ft 6 in. width clams is the very materially reduced crushing which ensues; and where prepared sizes are handled the reduction in degradation in all sizes from Nut through Lump or Block is considerable. There is one disadvantage with the light weight clams: in Block, Lump or Egg sizes they will not dig or fill to capacity as the heavier clams will do. In handling these sizes there is simply no substitute for weight. In the Mine Run sizes there is sufficient distribution of sizes so that the presence of the larger pieces does not interfere with the digging action of the clams. As with the slack and stoker sizes, the digging action of the clam depends upon the curve of the shell rather than upon the weight although excessive weight can make up for a poorly designed clam when handling the smaller sizes of material. The shape or curve of the back plate of the clam is as important for maintenance as it is in digging. One excellent instance of this is shown by the back shell plates of three of our clams where this piece is made of ½- to ¾-in. thick aluminum plate which has been in service for 10 years without replacement or maintenance of any sort and shows less wear than some clams with steel shell back plates. As would be expected, most of the wear occurs on the lip or bits which, if properly maintained with hard surfacing and occasional replacement with manganese wedge bars on the cutting

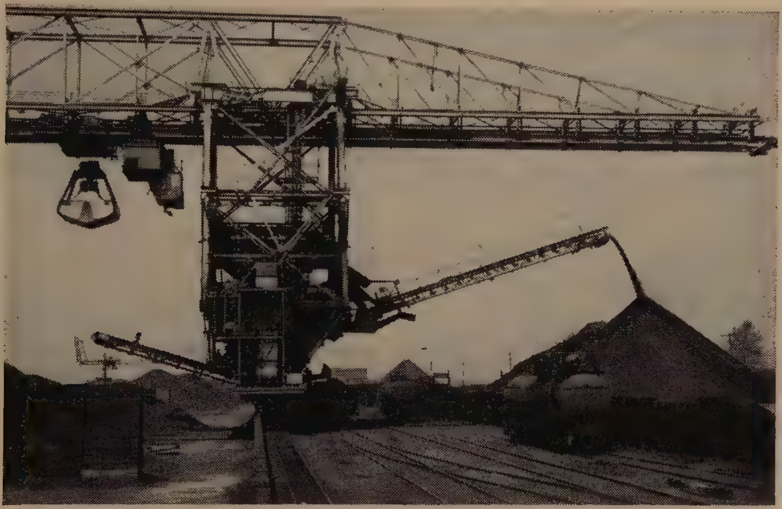


FIG 15—SCREENING PLANT IN BRIDGE PIER LEG, DOCK NO. 1.

edges, makes little difference as to the material of which the main section of the lips is made. Practically, a vanadium or nickel alloy steel or in fact ordinary AISI 1020 steel is perfectly satisfactory and relatively easy to maintain.

Screening Plants

In some instances, the screening plants are incorporated as a part of the bridge structure (see Fig 15). They are suspended between the legs of the pier or shear end of the bridge. In other instances, the screening plant is a separate unit attached to the bridge leg and rolling on the same rails but supported on its own wheeled truck structure. In general, where the screening plant is a portion of the bridge structure or is a separate structure, the operation consists of picking up Cargo or Dock Run sizes, screening out the largest size, which is then loaded, while the resultant undersize or Screenings is redeposited on the dock. This screen undersize is then picked up and run through the screening plant which in the meantime has changed its screening plates to the next smaller size. This operation may be continued as many times as there are desired sizes. The consequent breakage or degradation may well be imagined. Some screening plants are equipped so that two or three sizes may be recovered at the same time; although only one size may be loaded, the other sizes are stored in small 50 to 60 ton bins incorporated as a part of the screening plant. This is a convenient arrangement provided the sizes recovered may be balanced against the tonnage of sizes to be loaded and shipped. Such a situation is ideal but seldom achieved and it generally means that the sizes deposited in these small storage bins are eventually dumped back on the dock and recovered with additional screening at some future time. It is, however, an improvement over the first method as the tonnage to be rehandled each time is considerably less.

In recent years two installations, at Dock No. 1, Superior, and Dock No. 3 in Duluth, have been made which to a large extent have eliminated the rehandling of so much tonnage with the bridge spans. In these instances, concrete tunnels have been placed underneath the floor of the docks so that by means of small steel gates located in the roof of the tunnel, the coal may be drawn from the dock storage area through the roof of the tunnel onto belt conveyors running down the centers of the tunnels. The belt conveyors deliver the coal to modern screening plants located at the top of bin storage structures. In such installations, the screening process is virtually continuous as all sizes are recovered in one operation, each size of each variety of coal being placed in separate bins. From the bottoms of these bins, the coal is drawn out into trucks or onto conveyors which deliver it to a central car loading point. Any degradation resulting from the storage of the coal in the bins is removed by a small auxiliary vibrating screen prior to loading of the railroad cars. The use of such combinations of tunnel and bin screening plants has made a marked reduction in the breakage or degradation and a definite increase in size realization. The extent of this degradation is almost unbelievable in some coals. Such coals as the Pocahontas varieties, which may leave the mines as plus 7 in. Lump may have an average recovery off the docks of only 15 pct of the same size. This, then, means that 85 pct of the tonnage of this coal left the mines as Lump and is recovered from the docks and shipped only as the smaller sizes. Even the firmer structure Pittsburgh, Splint, and Elkhorn varieties leaving the mines as plus 5 in. Lump are recoverable off the docks at percentages between 40 and 60 pct of the original size. The rehandling of this degradation to recover the other sizes results in almost equivalent breakage. The tonnage of Lump sizes handled is steadily declining so that at the present time most of the effort is being made to increase the recovery of Egg, Stove and Stoker sizes from Cargo sizes. The advantages of the tunnel system over the present bridge handling systems will readily be realized when it is noted that the tunnel system handles the coal once to recover three to five sizes whereas the bridge screening plants normally require as many handlings as there are sizes to be recovered.

The only additional handling required with the tunnel plant system is the movement of coal from the dock storage area to the space above the tunnels. During the navigation season, this rehandling is seldom necessary as fresh cargoes are placed on the tunnel areas rather than over the entire storage area. Only the excess necessary for movement during the winter season is stored off the tunnels. By no means all of the degradation occurs on the docks as part of it is made when the coal is loaded into the cars at the mines, followed by breakage caused by vibration of the cars in transit, the loading of the lake freighters, and the movement of the vessels during their trip up the Lakes.

OPERATION

Probably one of the worst situations confronting the entire coal dock industry at the Head of the Lakes is the variety of equipment into which an even more varied number of sizes and kinds of coal must be loaded. Consignees may order their coal loaded into equipment for shipment over any one of seven different railroads. The particular kind of equipment may be gondolas, box cars, ore cars, or steel hoppers. The sizes comprise Block, Lump, Egg, Stove (2 sizes), Nut, Stoker, Run-of-Pile, and four sizes of Screenings from 2½ in. to 3⁄16 in. All of these variables, combined with 20 to 30 kinds of coal from Pennsylvania, Ohio, West Virginia, Maryland, Virginia, Kentucky and Tennessee, result in a number of operating and engineering problems which are frequently complicated by weather combinations of snow, 50 mile winds, and occasionally 30 below zero temperatures.

The following Table 3 of tonnage capacities of the docks is based solely upon experience, observations, and studies of tonnages handled by each unit over a period of the past ten years. It represents what is believed to be a logical balance between unloading and reloading capacities of each unit for the size and class distribution of coals presently utilized in the market area served by these docks.

TABLE 3—*Dock Capacities in Tons*

Unit	Theoretical Calc. Block Storage	Max. Economical Actual Dead Storage	Max. Annual Advisable Turnover	Peak Annual Possible Production
Dock 1	980,000	725,000	850,000	950,000
Dock 2	580,000	385,000	650,000	700,000
Dock 3	260,000	220,000	400,000	450,000
Dock 4	540,000	345,000	450,000	500,000
Dock 5	670,000	425,000	550,000	600,000
Total	3,030,000	2,100,000	2,900,000	3,200,000

Study of these tonnages shows what appears to be some rather obvious discrepancies, particularly on Docks No. 2 and 3. On Dock No. 2 the comparatively large tonnage is possible because of: (1) Type of coal handled—large piles of commercial screenings or slack or Mine Run; and (2) regular and predictable movement of 1200–1600 tons daily during most of the navigation season to one account via trucks all loaded by cranes thus freeing bridges for unloading and loading of other coals. On Dock No. 3 the tonnages shown are readily accomplished because: (1) A large proportion of retail and car lot shipments are handled through the tunnel thus freeing the bridges for unloading during navigation season and for rapid loading of direct loading coals; and (2) practically all coal screened is loaded out instead of being returned to the dock area for storage; in other words, shipments of sizes made in the tunnel plant are made to balance.

The approximate period April 10—

December 4 covers the extreme navigation season for the Head of the Lakes. During this period the bulk freighters carrying coal must be unloaded at the docks by the bridges and of course coal must be reloaded during the same period. Thus 24-hr days and 7 day weeks are not infrequent occurrences and are more apt to be the rule than the exception. December, January, and February are, like October and November, ordinarily months of heavy shipments and so the extreme weekly hours usually extend into those non-navigation months as well. March, April and May when dock stocks are well depleted, comprise about the only time available for major repairs. Hence, during this period occur the relaying or repairing of such dock flooring as can be laid bare in sufficiently large adjoining areas and any major repair jobs on the bridge structures and screening plants. During the latter part of April and all of May, cargo unloading is apt to interrupt repairs; but as reloading then is not much of a factor, heavy maintenance can generally be scheduled without too much interference.

Ordinarily it is known at the docks two or three days in advance what kind and size of coal will actually be loaded into a vessel and in which compartments each kind is to be placed. Loading of the vessel is to a considerable extent governed by the dock storage layout plan. Since vessel dispatch is important, the way a boat is loaded has a great deal to do with the rate at which it can be unloaded. By the same token the layout of the dock area at the beginning of a season is important since some coals ordinarily come forward in compartment lots and others as full cargoes.

Once the time of clearance of a vessel is known it is followed up the Lakes and estimates made of the arrival time so that other dock operations can be properly scheduled. Dependent upon the particular vessel we ordinarily allow 84 hr from Ashtabula or 72 from Toledo to Duluth-Superior. A check is made on the vessel when it locks through the "Soo" which is ordinarily considered 36 hr away. The last contact prior to docking is made whenever possible by radio phone when the vessel is from 2 to 8 hr out at which time any forseen peculiarities with respect to the unloading are discussed with the Master of the vessel. Each dock is furnished with a daily copy of the Vessel Situation Report (See Table 4) and with "spot" information as it is received so that any necessary alterations to man-power shifts and to loading schedules can be made. Depending upon the storage plan, the way the vessel was loaded and other factors of dock operation, at a dock which has two or more bridges, both bridges may start unloading and only one finish or vice versa. Very little man-power is required during the free digging part of unloading. In this case the unloading shift consists of a hoister and an oiler per rig plus one straw boss. After sufficient has been dug out of one or more compartments to permit cleaning coal off the interior gangways, walks or tank tops, and depending upon the detail of the interior construction of the vessel, a clean-up gang of two to eight men may be required. At this stage of the unloading one bridge may change from a digging bucket to a clean-up bucket with larger spread and in fact where the trolley is not equipped with a turntable, this change is almost mandatory. Some operations find that a small bulldozer of D-2 or D-4 size equipped with rubber or wooden fill-in pads to provide traction on the steel bottoms or pneumatic tired industrial tractors equipped with hydraulic scoop shovels or buckets are able to clean up the scattered heaps and ridges left by the digging clams and by pushing this underneath the hatch openings enable the digging clams to get much more of a "pay load" during the "clean up." The small tractors are easily placed in boats through the hatch openings by use of slings attached to the clam buckets. Where these means

TABLE 4—*Daily Vessel Situation North Western-Hanna Fuel Co. Duluth-Superior Docks*
October 18, 1947

Vessels in Port

Cargo No.	Name	Kind		Dock No.	Arrived	Started	Finished
227-4-52	E. T. Weir	Beckley M/R	Cpt 2)	4	9:30 PM 10/16	9:30 PM 10/16	9:30 AM 10/17
		Sewell M/R	Cpt 4)	4			
227-5-50		Island Creek 2" N/S	Cpt 1)	5	10:45 AM 10/17	11:00 AM 10/17	11:20 PM 10/17
		Pee Wee 5 × 0	Cpt 3)	5			
228-2-80	Shiras	Anth. Silt	Full Cargo	2	7:45 AM 10/17	7:45 AM 10/17	8:30 PM 10/17
229-5-51	Wolf	Unloads Cleveland-Cliffs	Cpt 3)				
	(Hicwilco)	Millers Creek M/R	Cpt 1)	5	2:10 PM 10/17	12:20 AM 10/18	3:00 PM 10/18 (abt)
		Marne Block	Cpt 2)	5			
		Marne Stoker	Cpt 4)	5			
230-1-75	Heekin	Dorothy N/S	Cpt 1 and 2)	1	8:45 PM 10/17	8:45 PM 10/17	6:00 PM 10/18 (abt)
	(Stern 1st)	Sewell Slack	Cpt 3)	1			
231-4-53	Hubbard	Elkhorn M/R	Cpt 1 and 2)	4	8:20 AM 10/18	9:15 AM 10/18	11:30 PM 10/18 (abt)
		Esco Res.	Cpt 4)	4			
231-3-47		Cavalier Stoker	Cpt 3)	3	Midnight 10/18 (abt)		5:00 AM 10/19 (abt)
232-2-81	H. W. Smith	Anth. Silt	Full Cargo	2	10:30 AM 10/18	10:30 AM 10/18	1:00 AM 10/19 (abt)

Vessels Enroute

Name	Kind			Dock No.	Cleared	Due
Townsend (Stern 1st)	Fairmont 2 in. N/S		Cpt 1)	2	Lorain 1:30 AM 10/16	3 AM 10/19
	N. P. No. 8 M/R		Cpt 2)	2		
	Fairmont 2 in. N/S	fwd	Cpt 3)	2		
	Fairmont Stove	aft	Cpt 3)	2		
Yates	Patsy, Everglow, Pruden Stoker		Cpt 1)	4	Sandusky 8 PM 10/15	6 AM 10/19
	Red Clover Stoker		Cpt 3)	4		
	Elkhorn M/R		Cpt 4)	4		
	Blue Diamond Furnace		Cpt 2)	5		
Sylvania (Stern 1st at Dock No. 1)	Wheelwright Lump		Cpt 2)	1	Sandusky 3 PM 10/16	8 PM 10/19
	Sewell Slack		Cpt 4)	1		
	Slab Fork Slack		Cpt 1 and 3)	2		
Joliet	Powellton 2 in. N/S		Full Cargo	3	Toledo 1 AM 10/17	7 AM 10/20
Conway	Dorothy 2 in. N/S		Cpt 1)	1	Sandusky 9:10 PM 10/17	4 AM 10/21
	Indiana Egg	fwd	Cpt 2)	1		
	Red Clover Block	aft	Cpt 2)	1		
	Millers Creek M/R		Cpt 3)	1		
Willis King	Kenallen 2 in. Crushed M/R		Cpt 1 and 3)	1	Ashtabula 7 AM 10/18	2 PM 10/21
	Penna Lovol M/R		Cpt 2 and 4)	5		

Cargoes Scheduled

Name	Kind		Dock No.	To Clear	Date
Pollock	Anth. Silt	Cpt 1-3-4)	2	Erie	PM 10/18
	Anth. Stove	Cpt 2)	4		
Macoubrey	Dorothy 2 in. N/S	Cpt 1)	1	Sandusky	PM 10/18
	Dorothy Egg	Cpt 2 and 3)	1		
	Shamrock 2 in. Mod. Scgs.	Cpt 4)	1		
Berwind	Unknown cargo			Sandusky	AM 10/19
E. L. Ford	Unknown cargo			Sandusky	AM 10/20
Hartwell	Georgetown 2 in. Crushed M/R	Cpt 1)	1	Huron	PM 10/21
	Willow Grove Stoker	Cpt 2 and 3)	1		
	Acme ⅜ × 5 for G.N.	Cpt 4)	1		

are utilized fully, and the scheduling of men and auxiliary equipment is very important, the number of men on the clean-up gangs can be materially reduced and vessel dispatch is speeded up.

Table 5 shows the Vessel Dispatch standing of our five docks for the period November 1 to December 5, 1947 and for the 1947 season.

TABLE 5—*Vessel Dispatch at North Western-Hanna Fuel Co. Duluth-Superior Docks, November and December*—1947

Dock	Tonnage	Tons per Hour
2	59,110.20	451.81
1	176,449.65	419.62
5	139,442.65	413.58
3	65,566.25	399.06
4	55,557.40	398.26
Total—Average	496,126.15	416.04
Cargoes Received		53
Split Cargoes		8
1947 Season		
2	686,865.10	500.44
1	890,454.45	476.01
5	517,651.00	433.12
4	398,031.85	394.48
3	388,165.20	334.33
Total—Average	2,881,167.60	434.61
Cargoes Received		306
Split Cargoes		89

The "ton per hour" column does not give an absolute value of a dock's unloading capacity since this depends not only on equipment and type of vessel unloaded but on the kind of coal and how stored. Dock No. 2 used 2 bridges with 8 ton clams for unloading while Dock No. 1 used 2 bridges with a 14 ton and a 7 ton clam and Dock No. 3 used two bridges with an 8½ and a 6 ton clam.

Of the five docks only Dock No. 1 has what might be considered average dock storage since it handles not only a normal amount of kinds and sizes of commercial coals requiring screening but also a fair proportion of direct loading Mine Run and steam slack. Dock No. 2 handles very little commercial coal, some 90 pct of its tonnage being direct loading steam coal of Mine Run or slack size which can be unloaded as a dump and run proposition and requires little care in unloading on the dock. Dock No. 3 by contrast handles 70 pct of its tonnage as commercial coal of which much is in prepared sizes with 25 pct of the total tonnage being prepared Pocahontas sizes which being a very friable coal requires especially careful handling. The "tons per hour" shown in Table 5 for vessel dispatch rate includes time spent for lunch periods, lubrication of clams and trolleys and any mechanical or electrical delays or time spent in shifting bridges and vessels; hence it is an average rate of unloading from the time the vessel tied up and the booms were lowered until the booms were raised and the vessel was able to depart. The split cargoes noted in the table are those where a vessel is unloaded at one or more docks and hence a partial cargo rather than a full cargo for any one dock. Split cargoes not only waste vessel time but increase dock cost as well since a split cargo can seldom be unloaded by using more than one bridge and the clean-up period is extended to two docks instead of one.

Where two bridges are used in unloading only one rig at a time stops for lunch periods. During these intervals the opportunity is taken to lubricate the trolley and clam. Slight mechanical or electrical repairs or adjustments are also made during the same periods whenever possible so that the unloading operation is as nearly continuous as it is possible to make it.

During a 15 min. period, 17.5 cycles closely approximating that of Table 6 can be made readily. During such a cycle the coal is actually raised about 45 ft, racked 150 ft and dumped. Such a cycle would apply to slack or Mine Run coal where degradation results from dumping from a height onto an existing storage pile while unloading a boat or transferring coal on the dock from one location to another. It also applies to loading of any kind or size of coal since emptying of the bucket into the bridge screening plant feed hopper bin does not permit lowering of clam. The cycle will apply likewise to almost any kind or size of coal after the storage area is nearly filled.

TABLE 6—*Trolley Operation Cycle at Dock No. 5*

Typical Performance Cycle While Unloading Vessel or Loading Where Trolley Run Does Not Exceed 150 Ft. Clam Capacity 16 Tons (22 Cu Yd)

CUMULATIVE TIME IN MIN.	PERFORMANCE
0.00	Start of Cycle—bucket open on coal—start to close
0.09	Bucket closed with hold line starting to take strain
0.10	Rack motors engaged at this point (accelerating)
0.30	Holding and closing motors disengaged at limit of raising
0.37	Rack motors shut off (coasting)
0.40	Rack motors plugged (braking)
0.42	Clam bucket started to open
0.47	Bucket completely open to discharge load
0.51	Rack motors reverse (accelerating)
0.64	Start to lower closing line
0.65	Start to lower hold line with dynamic brake
0.67	Rack motors plugged (braking)
0.78	Rack motor stopped—power shut off
0.86	All controls in neutral—bucket open on coal—end of cycle

With dusty coal, where dust reducing sprays have not been added at the lower lake loading ports, it is not an uncommon practice during the unloading process to add water by means of the vessel's fire hoses to those coals which experience indicates exhibit unusual tendencies toward spontaneous combustion in storage. This performs several useful functions. It effectively kills the dust and hence speeds up the unloading; it reduces segregation in the piles of coal on the dock which will reduce tendency toward spontaneous combustion; and the presence of the additional moisture in the pile will use up the initial heat in the pile. The reduction of segregation in the pile is the most important. When the coal is damp the smaller particles tend to cling to the larger pieces. This greatly reduces the "roll" off the pile so that any damp coal, be it Stoker or Screenings, will end up in storage with a more even size consist than will dry coal. Unloading damp serves two purposes: the more even size consist of the pile permits shipments from the pile with less chance of complaints of excessive "fines," and it reduces the chance of firing in those coals which have a tendency toward spontaneous combustion.

On docks which are assessed at $5000.00 or more per acre on valuable water front it is useless to talk of storage in terms of piles 10 to 20 ft high—under such conditions storage piles must be 40 to 50 ft high if full advantage of the property is to be taken. Likewise where the market requires it, a dock may have many small piles of from 3500 to 15,000 tons per pile. Such piles even if of slack coal cannot be bulldozed and tamped or rolled down because of the additional area required where space is already at a premium. On commercial docks there are few opportunities to pack and roll coal adequately since a pile may be lessened or increased from day to day. Such packing with dozers as can be done between unloadings is certainly beneficial, particularly where the edges can be packed. Unfortunately most of the literature on coal storage and methods of reducing spontaneous combustion is confined to locations where plenty of space is available or where hundreds of thousands of tons of one kind of coal go into one pile. On commercial docks the two best ways of reducing fires in addition to those already mentioned are to ship sufficient tonnage to keep the pile opened up and hence cooling off, and at the end of the navigation season to bring up the coals which have the greatest tendency to fire. The latter is seldom feasible. Often the coals with the greatest tendency to firing in storage come up early in the season and generally are of the slower moving coals that do not get shipped until the following early spring. The Mine Run or Slack (unwashed) from the No. 8 seam in Ohio causes the most trouble from fires; this is closely followed by coal from the same seam mined in the Fairmont, W. Va. area, and coal from the No. 4 Hazard seam. Removal of a portion of the "fines," say at ¼ in., materially reduces the tendency to firing and most modified screenings store fairly well. Washed coal, particularly where the −20 mesh has been partially removed and not re-added, handles nicely for the same reason. Certain Pocahontas, Beckley, Sewell and Midvolatile coals will fire because of the oxidation rate. In this respect some of the lowest ash-low sulphur coals are just as apt to fire as the high sulphur-high ash coals. In general however, the lower the rank of the coal the more the tendency to spontaneous combustion. Once a pile fires, by far the best remedy is to get sufficient tonnage shipped to open up the pile and thereby gain enough space so that the remainder can be turned over to permit cooling off. Actually and practically there are very few serious fires; and probably the tendency to fire of certain coals keeps the dock operations alert and consequently the entire operation benefits.

Reloading is apt to be either an extremely simple or a highly complicated procedure. After a coal handling bridge has been moved over the designated pile of coal the trolley and clam pick up and carry the coal to either the pier or shear leg hopper or bin usually of about 50 ton capacity. At all docks the operation known as direct loading merely consists of drawing coal by chute or conveyor from this feed bin and placing it either in open railroad cars or feeding it to a box car loader of the Mannierre or Ottumwa type. With the proper coal in the bin an open car can be loaded easily in from 3 to 4 min. and a box car in from 7 to 15 min. Where individual cars are to be loaded rather than a "string," much time can be wasted in getting the proper amount in a car since an order for an individual car can be anything from 32 to 50 tons. A bulkheaded car might contain 20 tons of one kind of coal in one end of a box car and 15 tons of another kind in the other end. During the winter months when ore cars are frequently used the situation is even more acute since a 50 ton capacity ore car holds only 20 tons of coal.

TABLE 7—*Sizes of Coal Ordinarily Prepared by Coal Docks at Duluth-Superior*

Size	High Volatile	Low Volatile
Block..............	Plus 5 in.	Not made
Lump..............	Plus 2 in.	Plus 2½ in.
Egg..............	2 × 5	3 × 8 (Cargo top)
Stove..............	1¼ × 2	1¼ × 3 and 1¼ × 2
Nut..............	Not made	⅝ × 1¼
Stoker..............	1 × ¼ (Cargo)	⅝ × ¼
2 in. Screenings.....	0 × 2	0 × 2 or 2½
1¼ in. Screenings...	0 × 1¼	0 × 1¼
⅝ in. Screenings....	Not made	0 × ⅝
¼ in. Screenings....	Not made	0 × ¼

When loading prepared or screened coal the procedure is somewhat more complicated and can best be illustrated by a description of the operation on several docks:

On Dock No. 1 at Superior, Wis., handling about 900,000 tons annually, the coal to be screened is drawn from the same hopper or bin in the pier leg of No. 1 bridge as may be used for direct loading and is fed to the shaker screen by a 48 in. pan conveyor. (Fig 16). The partially double decked screen makes any three product separation as shown below if Block or Lump is the size being loaded.

Separation	5 In. Block	3 In. Lump	2 × 5 Egg	1¼ × 5 Egg-stove	3 × 1¼ Stove	2 × 1¼ Stove	2 In. Scgs	1¼ In. Scgs
A	X		X				X	
B		X			X			X
C	X			X				X

For any of the above separations the Block or Lump must be loaded and the other two products may be returned to the dock. Ordinarily the 2 in. or 1¼ in. Screenings are elevated in a chain and bucket elevator and deposited by the belt conveyor boom out to the cantilever section of the dock, while the Egg or Egg and Stove are placed back on the main section of the dock via a steeply inclined pan conveyor operating as a hinged boom. If it is now or later desired to load either the Egg or Stove size the screen plates are changed and the Dock-Made Egg or Egg-Stove or Cargo Egg (if any) is clammed into the hopper. If Egg is desired it is loaded and the Stove size may be returned to the main dock and the 1¼ Screenings to the cantilever section; or the Stove and 1¼ Screenings are returned together to the cantilever (or main dock) as 2 in. Screenings. If Stove is required for loading it can be screened from Dock-Made Stove, Cargo Stove, 2 in. Screenings or from the Egg-Stove in which case the 2 × 1¼ in. is loaded; the 2 × 5 is returned to the main dock and the 1¼ in. Screenings are delivered to the cantilever section.

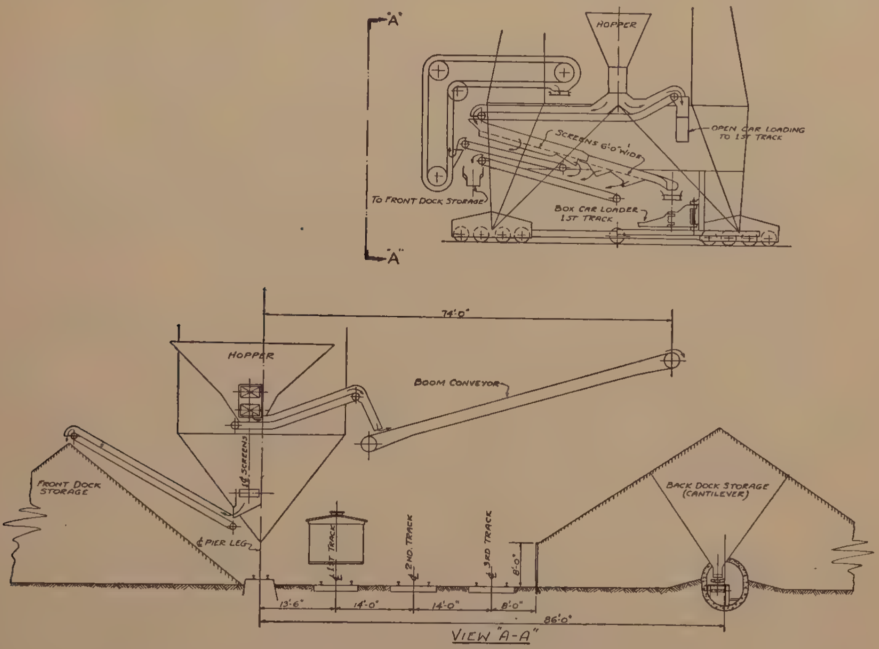


FIG 16—SCREENING PLANT FLOWSHEET. NO. 1 BRIDGE. DOCK NO. 1.

The coal deposited on one 750 ft long portion of the cantilever section of Dock No. 1 as 2 in. or 1¼ in. Screenings can be separated into 2 × 1¼ in. Stove and 1¼ × 0 Screenings or into 1¼ × ⅝ Nut, ⅝ × ¼ Stoker, and ¼ × 0 Slack if desired. Under this section of the dock is located a 7 ft high × 6 ft wide oval concrete tunnel with roof openings spaced every 7½ ft. Through these openings (See Fig 17) the coal stored above may be drawn and placed on a 30 in. belt conveyor 750 ft long which delivers to a 5 × 14 ft double deck vibrating screen. By changing the top deck screen cloth the above listed sizes can readily be made as the undersize of the bottom deck (¼ × 0) is always conveyed from the screen by a stacking conveyor. When making a three product separation one of the other two sizes is conveyed to bins of 75 ton capacity; the other size is conveyed to a loading house equipped with direct loading chutes, a box car loader, and a winch type car puller. This tunnel and screening plant, although originally erected as a pilot plant to test tunnel recovery, takes a big load off the bridge screening plant since it can screen and load coal while the bridge is unloading vessels or is shut down because of high wind or is loading other coal.

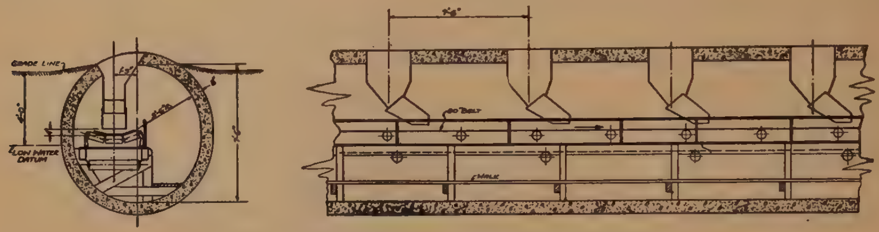


FIG 17—RECOVERY TUNNEL CROSS-SECTIONS.

The advantages of a cantilever section of a dock cannot be overestimated. In addition to providing additional storage area it primarily provides an area entirely removed (separated by railroad tracks) from the main dock storage area. Screenings derived as a byproduct of loading other sizes can be stored there as soon as recovered and kept available for direct loading without the inevitable addition or re-mixture of larger sizes which would occur if stored on the main dock area in close proximity to larger sizes. On Dock No. 1 an additional advantage of the stationary tunnel screening plant lies in the amount of coal handled through it. This can be dust treated by drawing from the 10,000 gal fixed tanks where otherwise large amounts of oil would have to be placed in the two 1000 gal tanks located in the bridge pier leg, which are too often empty when the bridge is 1000 ft or so away from its nearest spray oiling station. The tunnel plant requires a minimum of three men to operate: one man in the tunnel drawing coal, one working between the screen and the loading house, and one coopering cars and assisting in loading and working the car puller. Since each box car requires from 3 to 5 grain doors to prepare it prior to and during loading the 3 man crew may be increased to 4 if much of the shift loading is into box cars. The bridge crew required for screened coal loading consists of 8 men: a hoister, bridge oiler, screening plant oiler, loading foreman or rig boss, screen operator, box car loader operator, and two laborers preparing cars or cleaning up tracks. With the exception of the hoister, who is confined to the trolley, and the rig oiler, the remainder of the bridge loading crews work together on the various jobs and assist one another whenever possible.

On Dock No. 2 at Superior, Wis., which handles about 700,000 tons annually, two of the three bridges are used for "direct" loading of unscreened coal into open or box cars and the third bridge prepares and loads all of the screened coal shipped from this operation. The screening plant is located in a separate structure tied in with and attached to the pier leg and consists of a two section single deck shaker screen from which the oversize is loaded into railroad cars, an elevator which raises the undersize (generally −5 in. or −2 in.) to an overhead bin, and a double and a single deck vibrating screen.

Depending on whether the oversize of the main shaker for any particular loading is plus 5, 3 or 2 in., the vibrator screens can make a three size product separation of Egg (2 × 5), Stove (3 or 2 × 1¼), Nut (¾ × 1¼), Stoker (¼ × ¾) and 0 × ¼ Screenings. With any one set of screen cloths only two of the above sizes can be recovered and binned, and a third size can be returned to the dock. (See Fig 18.) The two bins available in the bridge leg for storing of the sizes from these screens hold approximately 50 tons and if the sizes balance within the limits of available orders each size can be loaded in turn. When this condition is not fulfilled the bridge must be moved a distance up or down the dock, usually not over 50 to 150 ft, to the appropriate storage location and the excess size or sizes are dumped on the dock via a conveyor and chute. When orders are available and the freshly screened coal in the bins is to be loaded, the screen plates on the main shaker are changed to the appropriate size and the coal from the bins is re-screened prior to loading to remove any degradation made in the binning and withdrawing process. Dock No. 2 is a relatively inefficient low tonnage plant and consequently is ordinarily restricted to the point where it handles only about 15 pct of its total tonnage as coal that is necessary to screen. Of an annual tonnage of 700,000 about 350,000 tons is loaded into trucks by cranes; 250,000 is reloaded by the bridges largely as industrial fuel and about 100,000 tons is prepared through the bridge screening plant. This dock has no cantilever section and if required to perform the same amount of screening as the other docks it would be necessary to reduce the annual tonnage materially or end up with many piles of coal of mixed sizes and varieties. The tonnage handled over this dock area would not be feasible were it not for the high percentage reloaded into trucks by cranes. This dock has an associated retail coal yard into which coal is transferred from the dock by railroad cars and clammed into overhead bins for truck loading of prepared sizes and into ground storage for conveyor loading of steam coals.

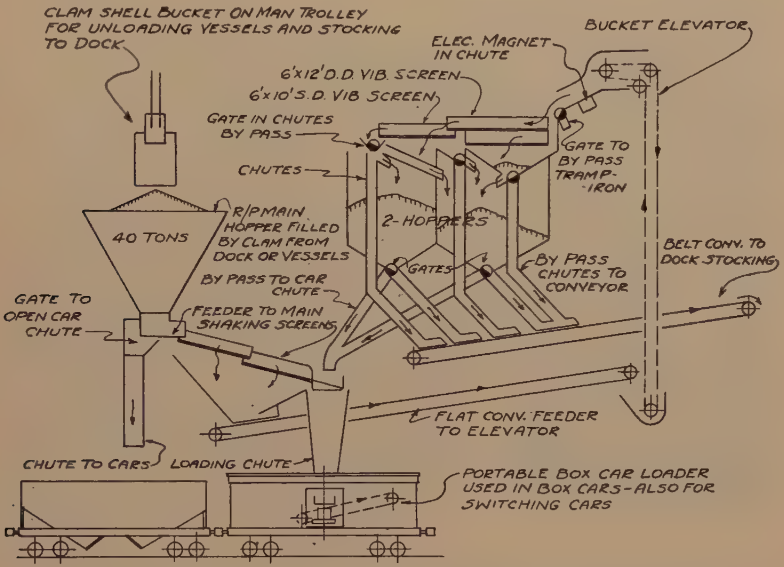

CLAM SHELL BUCKET ON MAN TROLLEY FOR UNLOADING VESSELS AND STOCKING TO DOCK
BUCKET ELEVATOR
ELEC. MAGNET IN CHUTE
6'x12' D.D. VIB. SCREEN
6'x10' S.D. VIB SCREEN
GATE IN CHUTES BY PASS
GATE TO BY PASS TRAMP-IRON
CHUTES
R/P MAIN HOPPER FILLED BY CLAM FROM DOCK OR VESSELS
40 TONS
2-HOPPERS
BELT CONV. TO DOCK STOCKING
GATES
BY PASS CHUTES TO CONVEYOR
BY PASS TO CAR CHUTE
GATE TO OPEN CAR CHUTE
FEEDER TO MAIN SHAKING SCREENS
FLAT CONV. FEEDER TO ELEVATOR
CHUTE TO CARS
LOADING CHUTE
PORTABLE BOX CAR LOADER USED IN BOX CARS - ALSO FOR SWITCHING CARS

Fig 18—Screening plant flowsheet. No. 1 bridge, Dock No. 2.

On Dock No. 3 at Duluth, Minn., handling about 400,000 tons annually, about 150,000 tons is reloaded direct by the bridges of which only one reloads at a time (both may be used for unloading vessels). About 250,000 tons is recovered via a 1000 ft long tunnel and belt conveyor located under the dock floor (Fig 17 and 19) and elevated by the same belt conveyor to the top of a large wooden bin structure made up of a double row of 16 bins: On top of these 32 bins the −8 or −5 in. coal is delivered to three 4 × 14 ft double deck flat vibrator screens which screen the low volatile coal into 3 × 8 in. Egg; 1¼ × 3

Stove; ⅝ × 1¼ Nut; ¼ × ⅝ Stoker and 0 × ¼ Screenings. When running high volatile coal it is screened into 2 × 5 Egg; 1¼ × 2 Stove and 0 × 1¼ Screenings or into 2 × 5 Egg and 0 × 2 Screenings. No lump coal is prepared over this dock as it cannot be fed consistently through the openings in the tunnel roof. One bridge is equipped with a 5 × 8 ft single deck vibrator screen which while most inefficient can be used on occasion to reduce Mine Run coal, which is usually handled on the section of the dock not served by the tunnel, to Lump and Screenings. During navigation season coal is clammed directly out of the boats onto the tunnel and the excess is stored behind the tunnel from where the bridges transfer it over the tunnel as required.

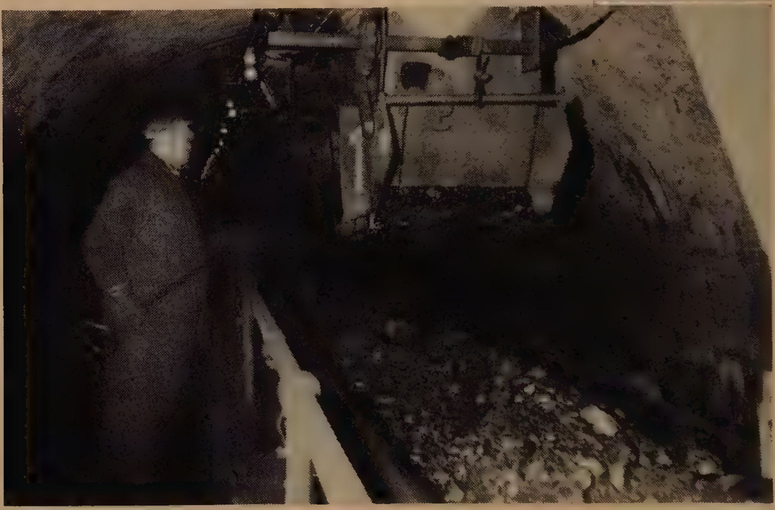


FIG 19—TUNNEL INTERIOR, DOCK NO. 3.

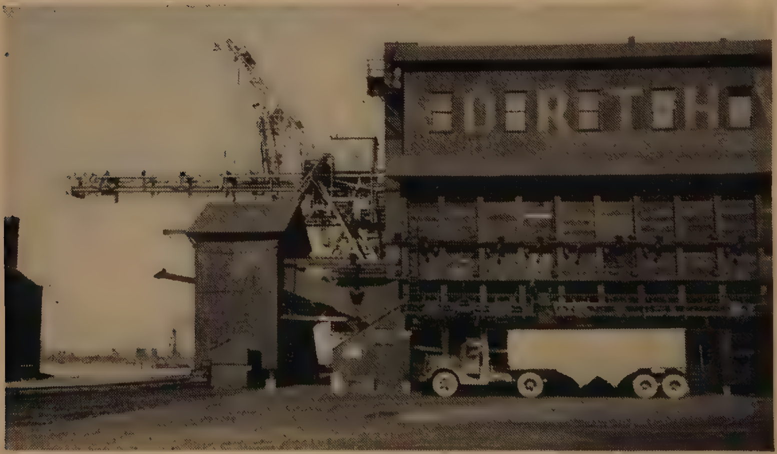


FIG 20—TUNNEL BIN SCREENING PLANT STRUCTURE, DOCK 3. CAR LOADING POINT AT LEFT. THE TRUCK HANDLES 26 TONS.

Because of the reduced percentage of degradation in coals handled through the tunnel and its associated screening and bin plant this dock handles the most friable Pocahontas and as much high volatile coal as possible where the varieties are to be screened down to 1¼ Screenings. The double row of 16 bins permits truck loading from the outer sides of the structure and car loading is accomplished by a centrally located conveyor at the bottom of the structure into which coal can be drawn from 26 of the 32 bins (Fig 20). The remaining 6 bins are at one end of the structure and are used for storage of Anthracite sizes, Coke, and Briquets which are received in railroad hopper cars and are delivered into these bins by a track unloading conveyor and an elevator. The car loading conveyor delivers coal from any one or any combinations of 26 bins to a car loading point which houses a small vibrator screen for any necessary removal of any degradation made in the bins or car loading conveyor. The oversize of this screen is loaded direct into open or box cars and the undersize is re-elevated to the top of the bin screening plant structure.

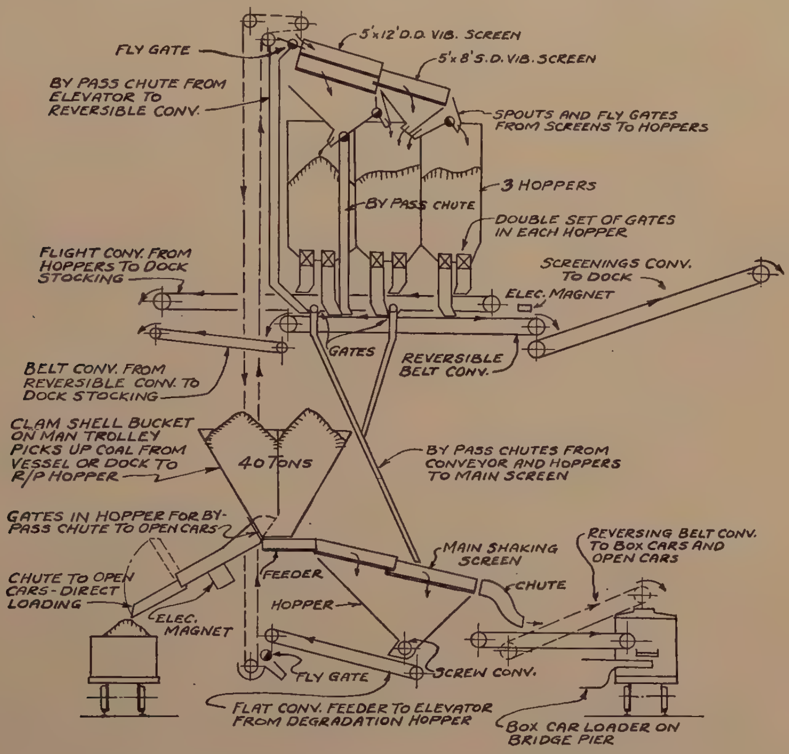

5'x12' D.D. VIB. SCREEN
5'x8' S.D. VIB. SCREEN
FLY GATE
BY PASS CHUTE FROM ELEVATOR TO REVERSIBLE CONV.
SPOUTS AND FLY GATES FROM SCREENS TO HOPPERS
3 HOPPERS
BY PASS CHUTE
DOUBLE SET OF GATES IN EACH HOPPER
FLIGHT CONV. FROM HOPPERS TO DOCK STOCKING
SCREENINGS CONV. TO DOCK
ELEC. MAGNET
GATES
REVERSIBLE BELT CONV.
BELT CONV. FROM REVERSIBLE CONV. TO DOCK STOCKING
CLAM SHELL BUCKET ON MAN TROLLEY PICKS UP COAL FROM VESSEL OR DOCK TO R/P HOPPER
40 TONS
BY PASS CHUTES FROM CONVEYOR AND HOPPERS TO MAIN SCREEN
GATES IN HOPPER FOR BY-PASS CHUTE TO OPEN CARS
MAIN SHAKING SCREEN
REVERSING BELT CONV. TO BOX CARS AND OPEN CARS
CHUTE TO OPEN CARS-DIRECT LOADING
FEEDER
CHUTE
HOPPER
ELEC. MAGNET
FLY GATE
SCREW CONV.
FLAT CONV. FEEDER TO ELEVATOR FROM DEGRADATION HOPPER
BOX CAR LOADER ON BRIDGE PIER

FIG 21—SCREENING PLANT FLOWSHEET. PIER LEG DOCK NO. 4.

The combination of car loading and truck loading from this plant plus the 100 ton capacity bins for each kind and size handled ordinarily balance out very well so that only a relatively small tonnage is returned to the dock either by truck or by car. About 150,000 tons is hauled annually by trucks from this plant for delivery within the city of Duluth and surrounding area. These trucks range in size from five to twenty-five ton capacity. The remainder of the tonnage drawn through the tunnel or about 100,000 to 150,000 tons annually is loaded into railroad cars. An additional 100,000 to 150,000 annually is loaded direct by the bridges.

Dock No. 4 at Duluth, Minn., has one bridge which handles between 450,000 and 500,000 tons annually. The screening plant in this bridge is closely patterned after that in the bridge leg at Dock No. 2 (See Fig 21). Coal from the loading hopper flows to a single deck shaker screen from which the oversize can either be loaded into any type car or can be returned via conveyor to the dock. Undersize of the shaker screen is elevated to a small bin which feeds a 5 × 12 ft double deck and a 5 × 8 ft single deck vibrating screen.

This plant is provided with three 50 ton capacity bins so that three sizes can be held and successively loaded if available orders balance the size recovery. In addition any size as made or after binning can be transferred via a belt conveyor boom to the cantilever section of the dock. As a result of the combination of the cantilever, three bins in place of two, and a higher elevation of the shaker screen oversize discharge lip, this plant has almost twice the hourly capacity as that of the plant on Dock No. 2 with nearly identical pieces of equipment. This dock is relatively short and as loading can be done at one end of the bridge only it provides limited railroad track loading facilities and thus does not lend itself well to bulk loading of large tonnages as do Docks No. 1 and 2. Hence the storage areas of this operation are used mainly for coals requiring screening.

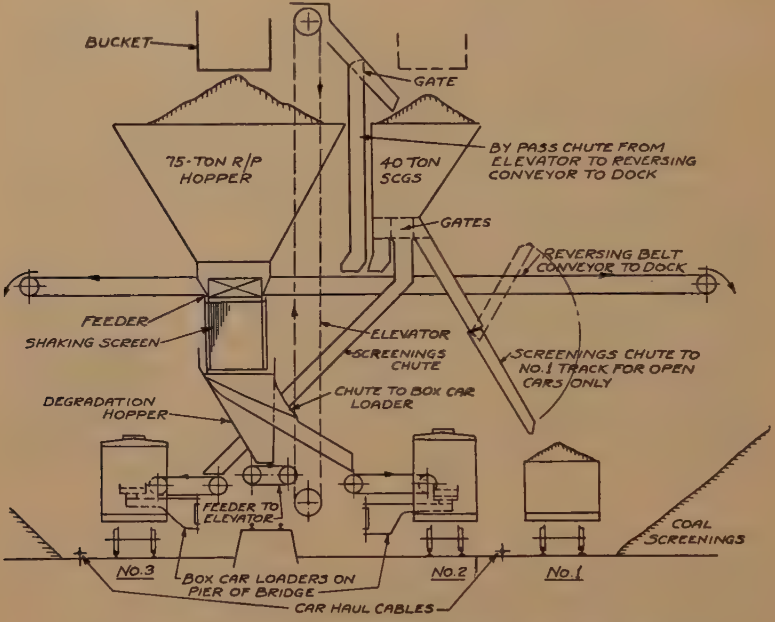


FIG 22—SCREENING PLANT FLOWSHEET. BRIDGE PIER LEG DOCK NO. 5.

Dock No. 5 like Dock No. 4 is a one bridge dock with a cantilever section and provides loading facilities at the pier end only of the bridge. Railroad tracks are located on each side of the pier leg foundation runway so that actually two loading points are available. The screening plant in the pier leg consists solely of a single deck shaker screen from which the oversize must be loaded into cars and the undersize, after elevating, can be returned either to the main dock or to the cantilever section. (See Fig 22). On this dock each size desired must be recovered in turn in descending order of size. If the 0 × 1¼ Screenings are to be recovered from Cargo Run of Pile Lump it is necessary to first screen out and ship the Block or Lump, then the Egg and finally the Stove size before the Screenings are available. Despite the excessive degradation caused by such a layout, the screening plant is capable of high tonnages and because of the double loading facilities this dock is well suited to loading both screened coal and large tonnages of direct loading steam coals.

Car supply for each dock is dependent upon a number of factors: The furnishing railroad, proximity of empty railroad yards, dock storage tracks, and method of handling cars on the dock. Each dock has engine service to bring in empties over the dock scale and set them on the dock tracks in the desired location in close proximity to the points of loading. Once the empties are placed, cars are moved into exact loading position by several means. Docks No. 1 and 3 use winch type car pullers located on the bridges or at the tunnel plant head houses to spot and load from 8 to 20 cars. These cars are then removed by the railroad locomotive while the bridges move on to new loading locations at which cars have previously been set. Dock No. 2 utilizes the self propelled box car loaders which run on their own tracks and are used to move cars into loading positions and to handle them during the loading process. Docks No. 4 and 5 are equipped with cable car hauls running the full length of the dock. By means of cable grips and short lengths of cable, cars can readily be spotted and moved past the loading points at any desired rate. By means of frequent push button stations the car haul cable can be started, stopped and reversed from convenient locations along the length of the loading tracks. Such a system is convenient and flexible but has been responsible for an excessive number of accidents.

Loading rates, as would be expected, vary greatly with the kind and size of coal, type of railroad car equipment, weather, and other conditions. For the same reasons, daily production may vary wildly not only between units but at the same unit from one day to the next. A bridge unit capable of loading Mine Run or Screenings into open cars at the rate of 800 tons per hr seldom is enabled to do such loading since orders for a particular type of coal never amount to that much for a day's loading and seldom enough for an hour's loading. The same bridge unit in loading Pocahontas Lump into box cars may load at the rate of one car every three hours if using Mine Run as feed to the screening plant or at 2 to 3 cars per hr if using Cargo Lump as a source of screen feed. If the screen feed consists of a hard structure high volatile coal which is resistant to breakage, the loading rate of plus 5 in. Lump may be 250–300 tons per hr into open cars or 100–200 tons per hr into box cars. Loading box cars (40 tons) with direct loading or unscreened coal may be as low as 160 tons per hr or as high as 400 tons per hr. Dock No. 1 has loaded as much as 6369 tons in one day following a 4653 ton day and preceding a 4530 ton day.

ELECTRICAL POWER

All power used at each of the five docks is purchased from the local utility companies at 13,200 volts, 25 cycles. At Docks No. 1, 4 and 5 this is changed to 600 volt dc, by synchronous rotary converters before entering the dock distribution system; and all motors and controls are rated for 550 volt. At Dock No. 3 transformers reduce the primary voltage to 470 volts before entering the dock distribution system and all motors and controls are rated for 440 volts at 25 cycles. At Dock No. 2 the power is distributed on the dock at 13,200 volts and picked up at that voltage on the shear end of each bridge. This full voltage is transmitted along the span to the pier end of each bridge where transformers reduce it to 440 volts, 25 cycles for use.

Dock No. 1

Dec. 31 1947

Recapitulation																	
Sheet No.	Pounds		Kind	Day, Tons		Month to Date, Tons		Kind	Day, Tons		Month to Date, Tons		Kind	Day, Tons		Month to Date, Tons	
	57	4	Champion-Mountaineer					Red Clover-Odin					Apex Poca				
	68	9	Scrd. Lump					Scrd. Lump			1838	55	Scrd. Lump				
862	419	5	Scrd. Egg	95	20	1140	10	Scrd. Egg			508	85	Scrd. Egg				
3	233	3	Scrd. Stove					Scrd. Stove			357	95	Scrd. Stove				
4	191	4						R/P 1″ Dom. Stkr.					Scrd. Nut				
5	96	5						2″ Scrgs.					Scrd. Pea				
6	214	5	R/P Lump					1¼″ Scrgs.			4596	90	Dom. Stoker				
7	220	8	R/P Egg	23	00	161	25						¾″ Slack				
8	101	7	R/P Stove	82	10	2908	00						5/16″ Slack				
9	257	8	R/P Dom. Stkr.	59	35	4078	95										
870	82	1	2″ Scrgs.			84	95										
1	222	4	2″ Stok. Scrgs.										Jewel Pocahontas				
2	93	9	1¼″ Scrgs.					Grenadier					Scrd. Lump				
3	197	9	Fm't Stoker	42	70	332	70	Scrd. Lump.					Scrd. Egg				
4	415	6	Pitco 1¼ Scgs.			155	75	Scrd. Egg					Scrd. Stove				
5	198	7						Scrd. Stove					Scrd. Nut				
6	413	1						2″ Scrgs.					1¼″ Slack			869	60
7	212	4						1¼″ Scrgs.					¾″ Slack				
8	75	3						Willow Grove Scgs.			1906	60					
9	522	8						Piney Fork Mod. Scgs.	214	00	5686	65					
880	89	7	GN Pgh. #8 M/R			3538	90	Blue Dragon-Dorothy					Penna. Low Vol. Scgs	159	60	4066	35
1	319	2	Railroad Fuel					Scrd. Lump									
2	229	0	G. N. Ry.					Scrd. Egg			1026	20					
3	220	8	Champ. Scgs.			279	55	Scrd. Stove									
4	77	5	Acme ⅜ X 5			383	65	2″ Scrgs.	461	20	7528	90					
5	180	2						2″ Stok. Scrgs.			234	85					
6	190	3						1¼″ Scrgs.					Sewell Poca				
7	105	7	N. P. Ry.					Shamrock N/S	114	50	1831	65	Lump				
8	188	6						Almer 2″ N-S					Egg				
9	400	9						Badger-Is. Creek					Stove				
890	351	2						Scrd. Lump					Nut				
1	108	2						Scrd. Egg					⅝″ Scgs.	160	25	1412	50
2	522	4	Oma. Ry.					Scrd. Stove					Cargo Scgs.				
3	197	3						2″ Scrgs.									
4	218	2						2″ Stok. Scrgs.									
5	93	5						1¼″ Mod. Scrgs.	332	70	4307	40	Proc. Smithing				
			Milw. Ry.					Dom. Stoker									
			Ohio #8 M/R			6441	80	Cavalier									
	7788	7	Georgetown N/S			3469	90	Scrd. Lump			1472	80					
								Scrd. Egg			2832	25	Total Our Coal	3050	65	81912	30

Recapitulation		Kind	Day, Tons		Month to Date, Tons		Kind	Day, Tons		Month to Date, Tons		Kind	Day, Tons		Month to Date, Tons	
Sheet No.	Pounds															
		Soo Ry.					Scrd. Stove			33	80	Handling				
							Dom. Stoker					Powellton	416	30	5651	80
							2″ Scrgs.					Dovey Scgs.	220	80	3656	70
							2″ Stoker Scrgs.					Kopperston	206	60	3558	25
		D. M. & I. R. Ry.					1¼″ Scrgs.									
		L. S. T. & T. Ry.					Georgetown Lump			134	10	Total Handling	843	70	12866	75
							Georgetown M/R					Total Bituminous	3894	35	94779	05
		M. N. & S. Ry.					Georgetown 2″ N-S					Anthracite				
							Georgetown Egg			116	10	Stove				
							Georgetown Egg R/P			42	25	Nut				
		Other Dock Co's.										Pea				
		Stott GN #8 M/R	898	65	3949	95						Buck				
		H. W. Milw. #8 M/R			99	75										
		Reiss Milw. #8 M/R			1367	75										
		Natona-Brunhild										Total Anth.				
		Scrd. Lump	75	60	5452	10						Coke				
		Scrd. Egg			2272	20						Stove & Nut				
		Scrd. Stove	94	30	1172	60						Pea				
		2″ Scrgs.			609	00						Breeze				
		1¼″ Scrgs.	237	50	3209	20	Ky. Kernel-Hazard					Total Coke				
		Stoker					Scrd. Lump									
							Scrd. Egg					Briquets				
							Scrd. Stove									
							2″ Scrgs.									
							1¼″ Scrgs.					Total Anth., Coke, Briq.				
												Grand Total	3894	35	94779	05

FIG 23—DAILY SUMMARY OF DOCK SHIPMENTS. NORTH WESTERN-HANNA FUEL COMPANY.

Weather: Clear-Cold

Date: 2-12-48

Dock No. 3

Operation	Tons
Unloading Cargo No.	
Loading	
Bridge Screened	
Bridge Direct	1101.95
Tunnel Rescreened	43.25
Tunnel Direct	447.95
Cranes	
Total Loaded (Cars)	1593.15
Transfer Dock (Truck)	50
Transfer Tunnel Plant	600
Transfer Fire	500
Total Transfer	1100
Boiler Fuel	
Electric Power Demand	3.5–30

Time

Operation	Hours	
Tunnel Plant Screening	14	
Tunnel Plant Loading	6	
Tunnel Plant Delays	None	
Bridges	No. 1	No. 2
Loading	5	
Transfer Tunnel	3	
Delays	None	
Transfer Fire	3	
Unconsigned Cars		

Delays

Remarks:
Fire in H.F. Screenings

Man Report

Class	Shifts 1st	2nd	3rd	Tot.
Bridges				
Bosses	1			1
Hoisters	1			1
Oilers	1			1
Screeners				
Loaders	1			1
Laborers	1			1
Tunnels				
Bosses	1			1
Screener	1	1		2
Oiler	1	1		2
Tunnel	1	1		2
Loader	1			1
Laborers				
Truck Loader		1	1	2
Cranes				
Operators				
Laborers				
Maintenance				
Foreman	1			1
Mechanics	1			1
Welders	1			1
Helpers				
Electricians	1			1
Helpers				
Laborers				
General				
Supervision	1			1
Technical				
Clerks				
Weigh.—Bill.	1			1
Fire—Watch		1		1
Laborers				
Janitor	1			1
Total	17	5	1	23

Yard No. 3

Operation	Tons
Unloaded Car Coke	
Loaded Our Trucks	278.45
Loaded Hired Trucks	71.45
Loaded Outside Trucks	613.30
Total Delivered	963.20
Obtained Other Yards	
Total Loaded Our Yard	
Returned to Yard Transfer	
Carried 14 %	40

Truck Report

Truck No.	Working Time	Disposition
257	9	Work
258	9	Work
259	9	Work
322	11	Work
331	10½	Work
336	10½	Work
337	9	Work
401	9½	Work
403	9½	Work
404	9	Work
405	9½	Work
414	10	Work
415	5	Work
417	10	Work
418	10	Work

Hired Trucks Used 8

Man Report

Class	Day	Night	Total
Bins			
Loaders	3		
Laborers			
Trucks			
Drivers	18½		
Helpers	8		
Maintenance			
Garage	1½		
Off	2		
General			
Supervision	1		
Weigh.—Bill.	1		
Laborers			
Total	35		

Truck Report

Truck No.	Working Time	Disposition
419	10	Work
420	10½	Work
421	9½	Work
424	10	Work
413		DL-Mount Conveyor

Fig 24—Daily operating log. North Western-Hanna Fuel Company. Lake Superior Division.

The following tabulation enables comparison of power consumption between dock units for the year 1947:

	Dock 1	Dock 2	Dock 3	Dock 4	Dock 5
Max. 15 min. kw demand	432	564	337	484	426
Kw-hr per ton shipped	1.55	1.38	1.60	2.07	1.89
Kw-hr per ton handled	0.78	0.92	0.80	1.04	0.95
Power cost per ton shipped	$0.021	$0.026	$0.030	$0.037	$0.032

OPERATION CONTROLS

Control of each operation is accomplished by close personal supervision supplemented by daily reports and monthly cost records. Cost sheets covering periods of less than half months would be virtually worthless since the day to day operation varies so much that comparison of units or within a unit is impossible. During the season of navigation a dock may easily go for periods of three or four days when no shipments are made off dock because of unloading activities; this might be followed by a like period of loading devoted mainly to screened coal loading followed by a period of bulk loading.

Operating under such schedules is best controlled by daily operating reports, two of which are shown. The Daily (and cumulative) Summary of Shipments (Fig 23) is used to govern distribution of loading orders to the docks and in conjunction with the Daily Operating Log (Fig 24) when coupled with close physical contact furnishes more accurate and satisfactory control than would be had with a daily cost sheet.

MAN POWER

Although kinds and tonnages of coal handled at each dock together with the equipment used vary to such an extent that direct comparison of operating units has little meaning, the tabulation of tons per man hour on an annual basis does yield some interesting figures.

Table 8 shows for the year 1947 and for the three year period of 1945–46–47 the tons shipped per man hour for the five North Western-Hanna Fuel Co. docks at Duluth-Superior and also comparable data for docks of seven other dock companies operating in the same area.

TABLE 8—*Man Hours per Ton of Coal Shipped for Various Coal Docks*

Operating Unit	1947 Tons per Man Hour	1945–46–47 Tons per Man Hour
NW-H Dock No. 1	10·4	9·3
NW-H Dock No. 2	8·3	7·1
NW-H Dock No. 3	6·7	5·9
NW-H Dock No. 4	7·7	7·2
NW-H Dock No. 5	7·2	8·0
Ave. NW-H 5 Docks	8·1	7·5
Other Dock Co. A	5·2	5·8
Other Dock Co. B	6·5	6·1
Other Dock Co. C	5·1	5·1
Other Dock Co. D	5·3	4·8
Other Dock Co. E	6·1	5·2
Other Dock Co. F	4·7	4·8
Other Dock Co. G	3·6	3·5

Man hours for the North Western-Hanna Docks used in the above table comprise all personnel at each operation including supervision and car weighers and billers and excepting only purely clerical workers at the General Dock Office.

ACKNOWLEDGMENT

The writer is indebted to Mr. J. R. Warner, President, North Western-Hanna Fuel Co. for permission to make the information contained herein available to the industry and for his helpful suggestions; and to Mr. R. C. Buck, President, R. C. Buck, Inc., Engineers, Superior, Wis., for his interest and helpful comments during the preparation of this paper.

The Use of Illinois Coal in the Production of Metallurgical Coke*

By Frank H. Reed,† Member AIME, Harold W. Jackman,† and P. W. Henline†

(New York Meeting, February 1948)

A summary of the consumption of coal in 1945 shows that the coke industry accounted for 17 pct of the total coal used. No substitute for coke and the blast furnace in the reduction of iron ore has gained sufficient prominence so far to be given serious consideration. However, while the same slot-type ovens are being built and used now as were in use a few years ago, the problem of obtaining suitable coking coals, both high- and low-volatile, has forced many operators to revamp coal-blending facilities.

Until the beginning of World War II, high-temperature coke plants were able to secure the required high-volatile and low-volatile coals from two or three seams and from three or four mines. Thus, many of our coke oven plants were able to operate with storage bins for two coals only. The variance in raw materials was not great, and uniformity of blend was a relatively simple problem. Since the beginning of World War II, many and important changes have taken place. Demand for all coals increased greatly. Production, though kept as near maximum capacity as possible, could not meet the demands for coal generally, and particularly for the various kinds of coal which have proved to be satisfactory for specialized uses; coals were allocated as nearly as possible in the order of greatest need; storage piling was reduced to the minimum necessary to keep plants in operation; uniformity of quality could not be maintained, and with this last, uniformity of composition of blends for coking could not be maintained. Whereas before the war producers of metallurgical coke could fill their needs from three or four mines, by the end of the war it was not unusual to find shipments to a single coke oven plant coming from as many as 30 different mines, in a single month.

Since many of the mines producing both high- and low-volatile coal for high-grade metallurgical coke are rapidly nearing depletion, the problem of seeking new sources and of blending a number of coals in each plant for the production of high-temperature coke is facing practically every producer in the United States at the present time, and is one which will probably continue for many years to come.

Midwestern byproduct coke ovens in the Chicago and St. Louis areas use annually from 12 to 15 million tons of bituminous coals which are transported 500 to 700 miles from the Appalachian coal fields of Pennsylvania, West Virginia, and eastern Kentucky. Approximately two-thirds of this coal is high-volatile bituminous.

The critical transportation problem confronting the nation in 1943, and the growing scarcity of the best Appalachian coking coals, prompted the Illinois Geological Survey to propose a research program in which would be studied the coking properties of blends of low-sulphur, high-volatile Illinois coal with the high- and low-volatile coals from the eastern fields. Such blends containing Illinois coal, if substituted for the all-eastern blends normally coked, would result in important transportation savings.

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* Released by permission of the Chief, Illinois State Geological Survey, Urbana. Manuscript received at the office of the Institute July 26, 1948. Issued as TP 2491 in Coal Technology, November 1948.

† Illinois State Geological Survey, Urbana, Ill.

To investigate this problem of producing metallurgical coke from Illinois coals, the Illinois State Geological Survey, through the University of Illinois, entered into a contract with the Office of Production, Research and Development of the War Production Board on July 1, 1943, for a six-month period. This contract was renewed January 1, 1944, July 1, 1944, and January 1, 1945. The contract terminated on June 30, 1945. Since this date, the project has been continued by the Illinois State Geological Survey under the sponsorship of the State of Illinois.

The first step in this program was the design and construction of a small-scale, slot-type oven in which coal blends could be carbonized under conditions approximating those obtained in commercial ovens. The coke produced under these conditions should have physical and chemical properties directly comparable to those of coke produced commercially from the same blend.

Such an experimental oven of 500-lb coal capacity was built. Its operation was standardized by carbonizing coal blends that were being used at the time in commercial ovens, and comparing experimental results with those from average commercial operation. Blends containing Illinois coals were then carbonized in the experimental oven and their coking properties were determined. This experimental oven was connected with the byproduct recovery train formerly used in our experimental work with the sole-flue oven. Tar and gas were collected and evaluated.

Early pilot oven tests indicated that the Illinois coals tested had plastic properties different from the eastern high-volatile coking coals normally used in coke production. This necessitated special studies on the technique of blending Illinois coals with coals from other areas, and laboratory tests involving plastic studies of both Illinois and eastern coals were made. Data obtained have been applied successfully to coal blending procedure in our pilot oven studies.

Early in the experimental program it became possible to cooperate with commercial producers of metallurgical coke who had an interest in using Illinois coal in their plants. Through these valuable connections, certain blends of Illinois coal, after preliminary pilot plant tests, have been carbonized in commercial ovens over extended periods of time. Their behavior in plant equipment, their yields of coke and byproducts, and the economics of their extended commercial use were studied. These cooperative studies have played a valuable part in carrying out this project.

PILOT PLANT COKE OVEN

Design and Construction

The primary objective in design of the experimental slot-type oven was to construct a unit which would duplicate essentially a small section of a commercial oven, and in which the process of coking coal would be controlled rigidly. Only in the width of the oven was an attempt made to duplicate any size dimension of a commercial oven. The average width of most commercial slot-type ovens ranges from 13 to 21 in. The actual width of the experimental oven is 14 in. The oven was designed so that it could be operated to give the same heat penetration (average width of oven in inches divided by coking time in hours) and final temperature as obtained in commercial practice. Fig 1 is a photograph of this oven showing coke being discharged.

A diagrammatic sketch of the oven showing detailed cross-sectional views from front and side is shown in Fig 2. As with all slot-type ovens, heat is applied from flues on both sides of the oven chamber ((1) Fig 2). The coal space in the oven chamber is 36 in. in length, 35 in. in depth, and holds approximately 10 cu ft of coal per charge.

The side walls (4) and floor (5) of the

oven are made of silicon carbide tile, 2 in. thick. Each side wall consists of a single tile, and the floor is formed from two tiles laid end to end with overlapping joint. The walls are anchored at the back of the oven and left free to expand vertically and horizontally. They are held in place at the top and bottom by the surrounding brickwork, and are further supported on each side by two rows of long firebrick (6) which touch the oven walls and are, in turn, strengthened by steel angles (7) running the full length of the outside walls of the oven.

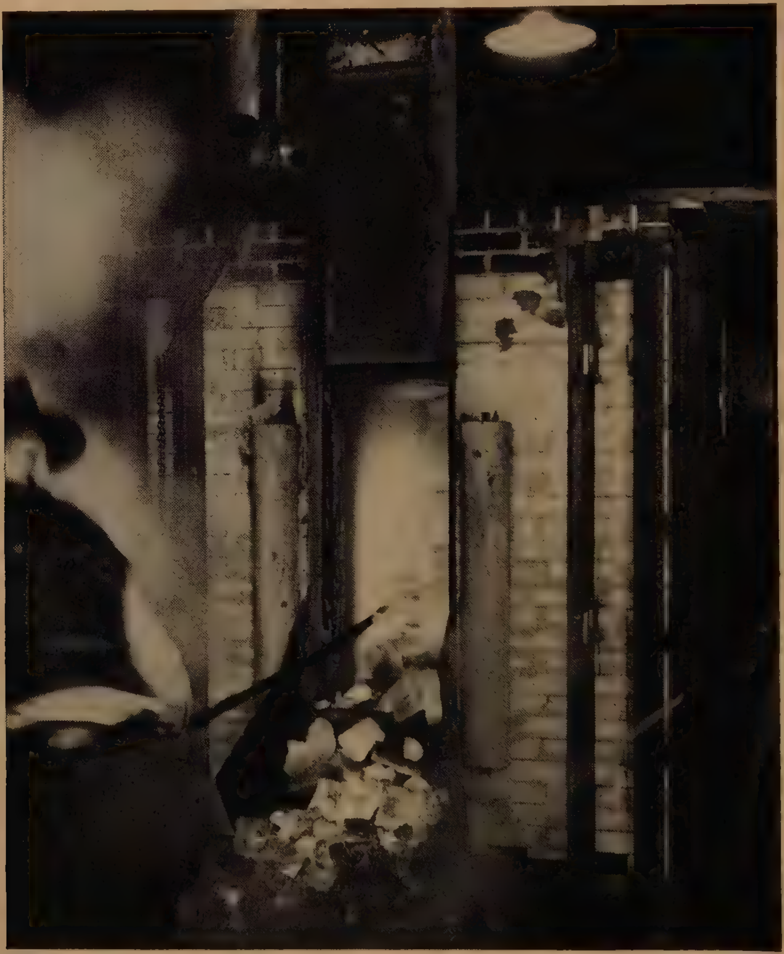

FIG 1—DISCHARGING AND QUENCHING COKE FROM EXPERIMENTAL OVEN.

The oven chamber is surrounded on the sides and top by vermiculite insulation (8). This insulation acts not only as a heat baffle, but, being soft, as a cushion against thermal expansion or swelling pressures which otherwise might crack the silicon carbide walls.

The top of the oven chamber (9) is cast of refractory concrete. Coal is charged through a 6-in. pipe (10) extending through

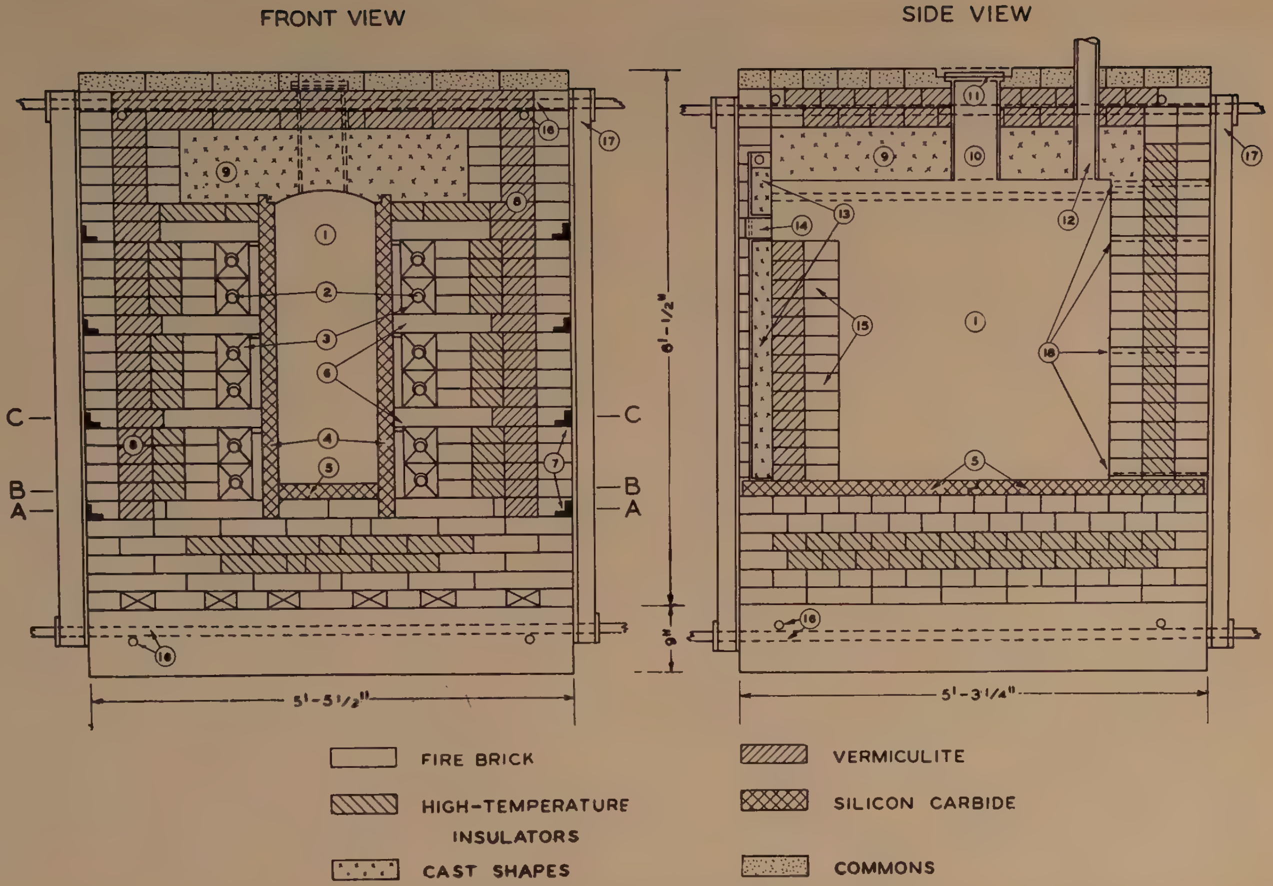

FIG 2—SKETCH OF SLOT-TYPE EXPERIMENTAL COKE OVEN.

the casting, and a 6-in. blank flange (11) serves as a charge hole cover. Gas escapes from the oven through a 3-in. pipe (12) extending through the top and connected to the byproduct recovery equipment. The back of the oven chamber consists of permanent brickwork, whereas the front is covered by a refractory concrete door (13) which is raised or lowered by a chain hoist and is mudded into place before the oven is charged. After charging, the coal is leveled through a rectangular opening (14) in the door located 35 in. above the chamber floor. This level bar opening is then bricked and mudded. Between the door and the coal charge a temporary brick wall (15) (9 in. in depth), extends from the floor to the coal level. This wall, which consists of one layer of firebrick next to the charge and one layer of insulating brick next to the door, is removed before a coke charge is pulled, and is replaced immediately after the oven is discharged. The oven structure is held together by tie rods (16) extending through the top brickwork and foundation. These rods are anchored to heavy buckstays (17) at each corner of the oven.

Temperature Control

Accurate control of the temperature and heating rate of the coal is maintained by regulation of Globar heating units which are powered from a three-phase 230-volt 60-cycle source through a 50 kv-amp tap transformer. Six AT type Globar brand nonmetallic heating elements ((2) Fig 2), 67 in. long and having a middle heating section 36 in. in length and 1.25 in. in diam, designed to carry a capacity load of 100 amp at 136 volts, are placed horizontally in each flue and spaced as shown so that heat may be applied uniformly from top to bottom of the oven walls. The two Globars in each top flue section are connected in series, and the two units thus formed are connected in parallel across one secondary of the transformer. Globars in the center and bottom flue sections are connected in a similar manner across the other two secondaries. In this way there are formed three independently variable single-phase circuits. Temperatures in the two flues are controlled separately by two Wheelco Capacitrols connected to thermocouples in the center flue sections adjacent to the oven walls. These units actuate the secondary circuits from the transformer. The even heating of the walls that is hereby obtained, together with the high heat conductivity of the silicon carbide tile, results in a very uniform application of heat to the oven charge; these factors are believed to be responsible for the uniformity of the coke produced.

Temperatures inside the oven are recorded by a four-point recorder actuated by thermocouples inserted through the back of the oven chamber ((18) Fig 2). Three thermocouples are located just inside the silicon carbide wall near the top, center, and bottom of the coal charge, and extending horizontally to the center of the oven. A fourth is placed in the exact center of the coal charge, and a fifth is located in the gas space above the coal.

Operation of this pilot oven has given coking results which check closely with average plant results. It is well known that because of difficulties in blending and heating coals uniformly, and the empirical methods of coke testing, no two consecutive ovens in a commercial battery yield coke showing identical physical properties. The close control of temperature in the pilot oven, combined with hand blending of the coals, has enabled us to check plant averages of one month or longer duration. One such check is shown in Table 1. It is noted that the size of the pilot oven coke is larger than that of the commercial oven coke. This is true necessarily because pilot oven coke does not receive the rough handling given to commercial oven coke in being pushed from the oven into the hot car, and passed through the plant screening equipment. Trends in coke size are shown accurately however in the pilot oven coke.

TABLE 1—*Comparison between Coking Results in Commercial Ovens and Pilot Oven on Same Coal Blend*

	Commercial Ovens 57 Days Average	Pilot Oven Run No. 102
Coke analysis, pct		
Volatile matter	1.5	1.0
Fixed carbon	87.4	87.8
Ash	11.1	11.2
Sulphur	0.77	0.69
Coke yields, pct of coal		
Total	65.0	65.0
Furnace (+1 in.)	58.1	62.2
Nut (1 × ½ in.)	3.4	0.7
Breeze (−½ in.)	3.5	2.1
Coke screen test, pct of coke		
Total +4 in	8.4	6.4
Total +3 in	35.8	48.4
Total +2 in	78.3	87.2
Tumbler test		
Stability (+1 in.)	49.3	49.0
Hardness (+¼ in.)	67.9	66.8
Shatter test		
Pct of +2 in	66.0	63.6
Pct of +1½ in	94.9	96.4

CONCLUSIONS REGARDING COKING OF ILLINOIS COAL

As a result of the tests made with Illinois coals which, on the basis of chemical composition and immediate availability in quantity, are the most promising for metallurgical coke production, the following conclusions may be drawn.

1. Illinois No. 6 seam coal from the Franklin Co. low-sulphur area can be used continuously in blends with eastern coals in modern slot-type coke ovens for the production of coke which is practical for use in commercial blast furnaces. The extent to which Illinois coal can be used to replace eastern high-volatile coal for this purpose is dependent primarily upon the economics of each individual application. Experimental pilot plant tests and commercial full-scale operation have shown that up to 75 pct of this coal may be used satisfactorily.

2. Such use of Illinois coal in metallurgical coke plants of the Chicago and St. Louis areas does result in sizeable transportation savings.

3. Cokes of satisfactory physical and chemical properties can be made from blends containing up to 75 pct or more of Illinois No. 5 seam coal from the limited low-sulphur area in Saline Co.

4. Cokes with equally good physical properties can be made using other No. 5 seam coals of medium sulphur content from Saline and Williamson Counties. These coals and others similar to them are worthy of consideration as small percentage constituents of coal blends.

5. Illinois coal fines should not be used for coking. Fusain tends to concentrate in the fines, and the tendency to weather is increased by the large surface area. No lower limit on screen size, as prepared at the mine, has been determined, but in actual applications no size smaller than ⅜ in. has been recommended or used for coking.

6. Sized and cleaned Illinois coal can be safely stocked without hazard of spontaneous combustion.

7. Because of the extensive use of cleaning plants in the low-sulphur area, the coal shipped from this area is very uniform in preparation and composition, and coals from the mines of the various producing companies are interchangeable.

8. The bulk density of Illinois coal when charged to coke ovens is almost identical with that of eastern coals. However, as a result of the higher inherent moisture content of the Illinois coal, a correspondingly lower yield of coke is obtained.

9. In general, the low-sulphur Illinois coals tested in this program become less fluid during carbonization than do the higher ranking eastern high-volatile coking coals. Our tests have shown that the coke structure of an Illinois-Pocahontas coal blend may be improved by including a portion of a more fluid eastern high-volatile coal in the blend, or by substituting certain medium-volatile coals for the low-volatile Pocahontas coal that is normally used in production of metallurgical coke.

EXPERIMENTAL RESULTS

Now let us take a brief look at a few of the experimental studies conducted by means of the 500-lb slot-type oven. With this oven it is possible to study the effects of coal pulverization, flue temperature, coking time, weathering, coal cleaning, and the blending of any combination of coals under any ordinary set of operating conditions. The tables which follow, illustrating representative data which may be obtained, summarize the results of certain of these studies.

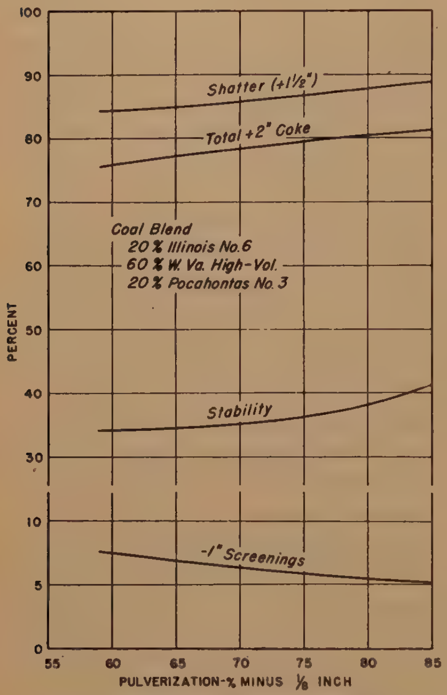

FIG 3—EFFECT OF COAL PULVERIZATION.

TABLE 2—*Effect of Coal Pulverization*

Pct
Coal Blend { 20 Ill. No. 6
60 W. Va. High-volatile
20 Pocahontas No. 3

Pulverization (pct minus ⅛ in.)	59	76	85
Avg. size of coke (in.)	2.75	2.88	2.93
Plus 2-in. coke (pct of total coke)	75.5	79.8	81.4
Minus 1-in. screenings (pct of total coke)	7.6	5.8	5.2
Shatter index (+2-in.)	64.7	72.7	71.4
(+1½-in.)	84.4	87.0	88.9
Tumbler test			
Stability	34.1	36.6	41.4
Hardness	59.4	57.8	62.9

As blast furnace coke accounts for the major coke tonnage produced and used in the Chicago area, the coal blends shown are those which either are used in production of coke of this type or which were being tested for such use.

PULVERIZATION STUDY

Experimental data showing the effect of coal pulverization on the physical properties of a typical blast furnace coke are shown in Table 2. In this series of tests the coal pulverization was varied as shown, and the bulk density of coal in the oven was maintained essentially constant by partial surface drying. This can be accomplished commercially by oil spraying.

Finer pulverization is shown here to produce a larger, stronger coke, and to decrease the minus one-inch screenings with a corresponding increase in the furnace size coke produced. Results of these tests are plotted in Fig 3.

Not all coal blends follow this pattern when carbonized. In other series of tests made, the coke size was found to decrease with finer pulverization, but in every case coke stability increased consistently as the coal was more finely pulverized. The probable explanation for this increased stability is that finer pulverization produces more uniform blending of the coals, and inert particles present are made smaller and are more uniformly distributed in the coal mass.

VARIATION IN COKING TIME

Commercial coke ovens vary the rate of coking over a wide range depending upon the type of coal blend being coked, the demand for coke, and the coke properties desired. For example, foundry coke is usually made by carbonizing at a relatively low temperature for a long period of time to produce a large, strong coke capable of resisting breakage in the foundry cupola. Blast furnace, or domestic, coke may be made at a much faster rate since a smaller-sized coke is more desirable for these purposes.

Table 3 and Fig 4 show the effect of coking time on a blend containing a high percentage of Illinois coal. Long coking time is shown to produce large size coke with correspondingly high shatter index. Coke stability remains high denoting toughness, but hardness decreases with a corresponding increase in breeze. A very low hardness factor indicates a soft surface structure which is easily abraded on handling, with formation of excessive coke fines.

These trends have been substantiated commercially in Illinois on this type of coal blend. Long coking time produced a large coke with high stability but considerable pebblyness and breeze. Shortening the coking time resulted in production of a smaller coke, uniform in size, with smooth surface structure. Breeze production was reduced. The use of fast coking rates is thus indicated when a large percentage of Illinois coal is included in the coal blend being carbonized.

TABLE 3—*Effect of Coking Time*

Coal Blend { 75 pct Ill. No. 6
25 pct Pocahontas No. 3

Coking Time (hr: min.) (computed for 19-in. oven)..	16:35	18	20	22	24
Avg. size of coke (in.)	2.31	2.44	2.58	2.97	3.21
Plus 2-in. coke (pct of total coke)...	64.0	70.5	74.8	81.2	83.4
Minus ½-in. breeze (pct of total coke)	6.0	5.7	6.5	7.5	9.0
Shatter index (+2 in)	61.3	67.4	70.9	81.1	83.9
(+1½ in)........	85.0	86.5	89.1	92.2	92.4
Tumbler test					
Stability (+1 in.).	49.2	50.3	52.1	51.9	51.7
Hardness (+¼ in.)	66.9	66.7	63.7	61.7	58.1

SUBSTITUTION OF ILLINOIS COAL FOR EASTERN HIGH-VOLATILE

In the series of tests shown in Table 4 and Fig 5, Illinois coals from both the No. 5 and No. 6 seams have been substituted in increasing percentages for portions of the West Virginia high-volatile coal in a basic coal blend consisting of 80 pct West Virginia high-volatile and 20 pct Pocahontas coals. A fast coking rate of 1.05 in. per hr was used throughout the series of tests.

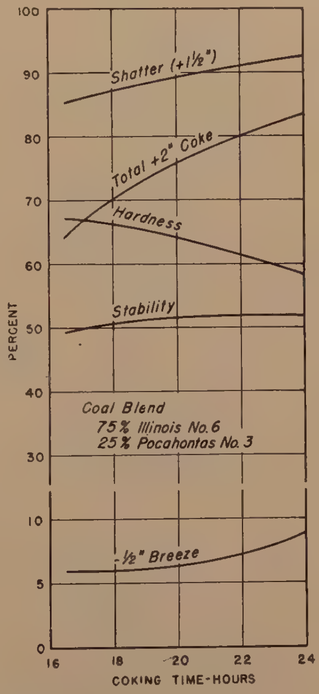

FIG 4—EFFECT OF COKING TIME.

It is seen that there was not much change in coke size except where a very high percentage of No. 5 seam Illinois coal was used. Breeze production likewise remained constant except when using 80 pct Illinois No. 6 seam coal. The outstanding effect of Illinois coal on the coke is to increase the tumbler stability and hardness. No. 5 seam coal is seen to be more effective in this respect than the coal from No. 6 seam.

WEATHERING OF ILLINOIS COALS

Illinois coals, which are of lower rank than the best eastern coking coals, are known to weather more rapidly than these eastern coals when stocked for extended periods of time. Weathering is accompanied by a reduction in coking properties, and may be shown by a series of coking tests made on the coal taken from stock at regular intervals of time.

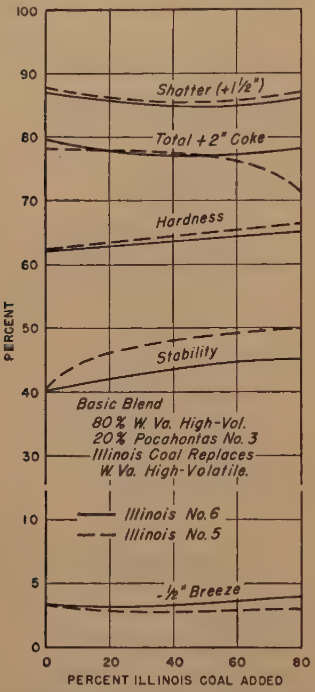

Fig 5—Substitution of Illinois coals for Eastern high-volatile coal.

Weathering occurs largely at the surface of coal pieces, where oxygen comes into direct contact with the coal. Elimination of coal fines reduces the exposed surface area greatly, and weathering of sized coal is much slower than that of run of mine or slack sizes. The following two tables show the effect of weathering on 2 × ⅜ in. washed No. 6 seam Illinois coal. This coal was stocked outdoors in small conical piles and sampled at monthly intervals. In each case a blend was made of 80 pct Illinois coal, 20 pct Pocahontas, and the blends thus formed were coked under identical conditions.

Table 5 and Fig 6 show the results of such a series of tests in which the Illinois coal was stocked during the summer months from April through August. It is seen that there was not much change in coking properties for the first three months of storage. During the fourth and fifth months, which were in the hottest part of the summer, weathering proceeded rapidly as shown by a decided reduction in the tumbler stability and hardness factors of the coke, and by an increase in the production of breeze.

Table 4—*Substitution of Illinois Coal for Eastern High-Volatile*

Part A—Illinois No. 6 Seam Coal

Basic Coal Blend {80 pct W. Va. High-Volatile; 20 pct Pocahontas No. 3

Pct Illinois coal added	0	10	30	50	80
Avg. size of coke (in.)	2.73	2.82	2.63	2.66	2.63
Plus 2-in. coke (pct of total coke)	78.5	80.7	76.6	76.4	78.7
Minus ½-in. breeze (pct of total coke)	3.4	3.2	3.5	3.4	3.9
Shatter index (+2 in.)	69.1	71.3	68.0	67.2	67.8
(+1½ in.)	87.3	86.9	85.3	84.6	86.1
Tumbler test					
Stability (+1 in.)	40.2	40.6	42.5	44.4	45.0
Hardness (+¼ in.)	62.1	62.1	63.5	64.1	65.4

Table 4 Part B—Illinois No. 5 Seam Coal

Basic Coal Blend {80 pct W. Va. High-Volatile; 20 pct Pocahontas

Pct Illinois coal added	0	10	30	50	80
Avg. size of coke (in.)	2.73	2.64	2.66	2.69	2.43
Plus 2-in. coke (pct of total coke)	78.5	77.8	77.4	77.6	71.1
Minus ½-in. breeze (pct of total coke)	3.4	3.0	2.9	2.8	3.0
Shatter index (+2 in.)	69.1	68.7	64.7	64.8	66.0
(+1½ in.)	87.3	88.1	85.4	86.2	87.1
Tumbler test					
Stability (+1 in.)	40.2	44.9	47.0	48.2	49.9
Hardness (+¼ in.)	62.1	65.1	64.1	64.9	66.4

Table 5—*Effect of Weathering (Summer)*

Coal Blend {80 pct Illinois No. 6 (2 × ⅜ in. washed); 20 pct Pocahontas No. 3

Months in storage	1	2	3	4	5
Avg. size of coke (in.)	2.43	2.42	2.51	2.49	2.37
Plus 2 in. coke (pct of total coke)	70.9	68.7	72.8	70.7	69.6
Minus ½ in. breeze (pct of total coke)	4.5	5.2	6.2	9.2	14.3
Shatter index (+2 in.)	63.3	71.8	63.5	65.4	69.2
(+1½ in.)	85.2	88.1	86.8	88.0	85.8
Tumbler Test					
Stability (+1 in.)	49.4	47.1	48.4	42.5	37.0
Hardness (+¼ in.)	67.2	66.5	64.8	59.8	51.5

Table 6 and Fig 7 show the results of a similar series of tests in which the Illinois No. 6 seam coal was stocked during the winter period, from September through March. It is noted here that although there was some variation in the results obtained, there was no definite reduction in the coking properties of the coal during storage. Coking properties were shown to be practically the same after seven months storage as after one month.

Consideration of these and other weathering test data obtained on washed and sized Illinois No. 6 seam coal indicates that where this coal is to be used in amounts as great as 80 pct of the total coal blend the Illinois coal may be stored for one to three months in summer weather, and for longer periods in cooler weather, without seriously affecting the coking properties of the coal blend.

The above tests represent the most drastic type of coal storage—that is, in small piles of from 2 to 4 tons where all coal is on or near the surface at all times. In actual plant storage piles of large size a much smaller percentage of the total coal surface is exposed to contact with air, and weathering is less pronounced.

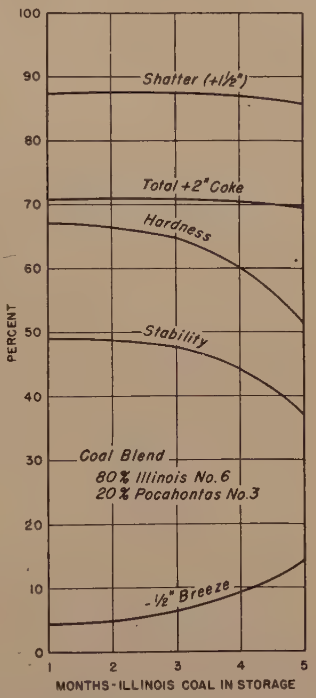

FIG 6—EFFECT OF WEATHERING (SUMMER).

TABLE 6—*Effect of Weathering (Winter)*

Coal Blend {80 pct Illinois No. 6 (2 × ⅜ in. washed)
20 pct Pocahontas No. 3

Months in storage	1	2	3	4	5	6	7
Avg. size of coke (in.)	2.45	2.50	2.57	2.56	2.56	2.57	2.58
Plus 2-in. coke (pct of total coke)	70.7	73.1	75.3	74.2	75.6	72.9	72.3
Minus ½-in. breeze (pct of total coke)	11.5	10.0	8.1	6.3	6.5	8.5	7.7
Shatter index (+2 in.)	68.0	68.6	65.9	70.6	71.0	69.9	68.8
(+1½-in.)	86.9	87.6	86.2	90.0	88.9	86.8	86.9
Tumbler test							
Stability (+1 in.)	44.1	43.0	43.8	46.6	46.5	45.5	43.8
Hardness (+¼ in.)	59.4	58.4	60.9	63.2	64.4	61.1	62.0

Tests were made on Illinois coal stored during the winter season in a ridge-shaped pile approximately 150 ft long and 25 ft high. Samples were taken from the interior of the pile after 1, 2 and 6 months of storage. The samples were blended as 25 pct of a blend of Illinois, Eastern Kentucky, and Pocahontas coals and coked in the experimental oven. Different samples of Kentucky and Pocahontas coals were used in each test which may account for minor variations in coking results, but, as shown in Table 7 and Fig 8, there was no continuous downward trend in coking properties during the six months storage period. It was noted also that the final Illinois coal sample removed from stock was still bright in appearance, showing no discoloration, slacking, or heating. Thus it is indicated that where washed, prepared sizes of Illinois No. 6 seam coal are to be used as not over 25 pct of the total coal blend the coal may be stocked safely over the winter months.

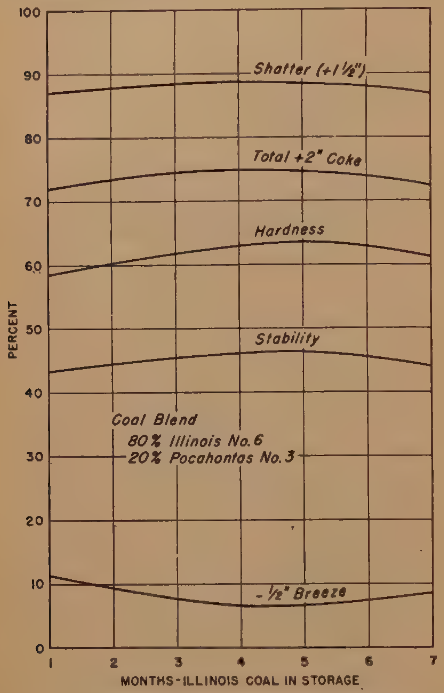

Fig 7—Effect of weathering (winter).

Coal Reserves

Table 7—*Weathering in Plant Storage (Winter)*

Pct
Coal Blend { 25 Illinois No. 6
25 Eastern Kentucky
50 Pocahontas No. 4

Months in storage	0	1	2	6
Avg. size of coke (in.)	2.55	2.55	2.63	2.63
Plus 2 in. coke (pct of total coke)	72.2	73.0	75.4	74.4
Minus ½-in. breeze (pct of total coke)	3.7	4.1	4.2	4.2
Shatter index (+2 in.)	61.5	64.0	62.2	62.9
(+1½ in.)	82.4	83.9	82.7	82.5
Tumbler test				
Stability (+1 in.)	36.0	31.3	28.3	34.5
Hardness (+¼ in.)	67.3	64.0	62.7	65.4

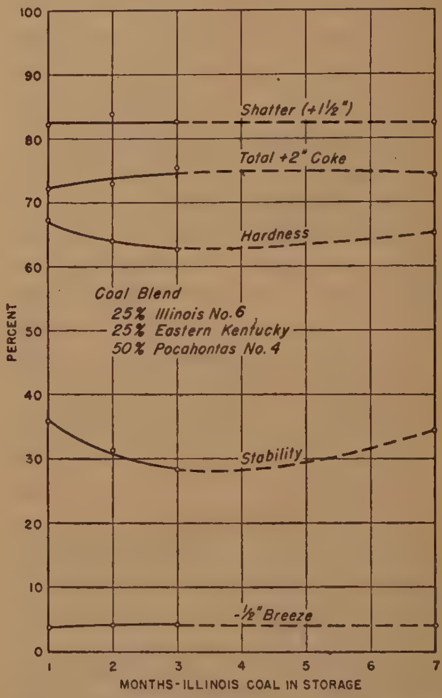

Fig 8—Weathering in plant storage (winter).

The question as to the amount of coal from any district that can be used for metallurgical coke is not relevant unless the amount of reserves are fairly well known. Consequently, an estimate of the amount of No. 5 and 6 seam coals in southern Illinois which could be used for metallurgical coke has been made by our geologists under the direction of Dr. G. H. Cady. A map of the area, showing the location of coal reserves and quantities available, is shown in Fig 9.

In making this estimate it was necessary to assume a maximum tolerance of sulphur content in the coal. While we have realized fully the lower the sulphur content the more desirable the coal for metallurgical purposes, it is well known also that the average sulphur content of coals used in coke ovens during the past few years has risen sharply in a number of cases. Considering this angle, and also the fact that Southern Illinois coal may be used to advantage in quantities from 10 to 25 pct of the total charge, and that in no case will Southern Illinois coal be the only coal used, a top figure of 2½ pct of sulphur content in the coal as it occurs in the mine was used

In most cases such a figure would result in a sulphur content of washed coal as shipped below 2 pct. This shows a coal reserve in Southern Illinois of approximately 3 billion tons, 60 pct of it in seam No. 6 and 40 pct in seam No. 5.

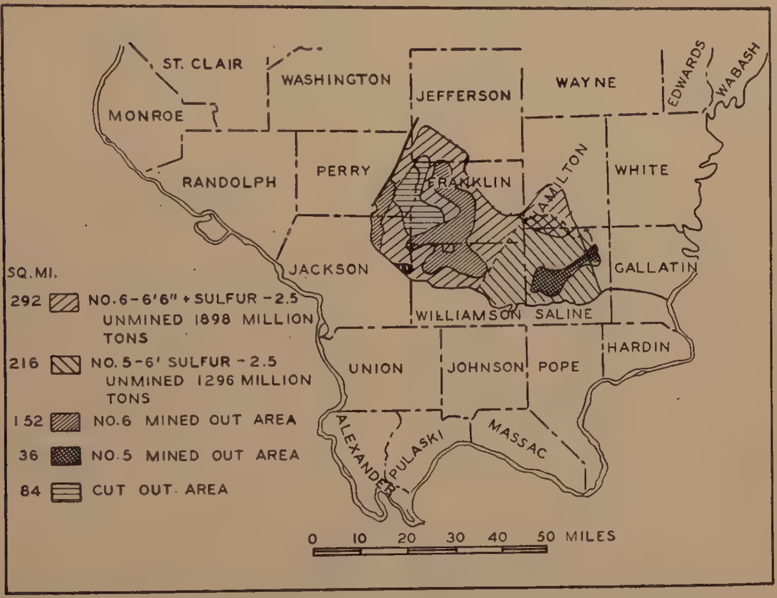

Fig 9—Map of southern Illinois showing low-sulphur coal reserves.

Conclusions

The conclusions which have been drawn above, based upon work in our experimental slot-type oven, are indicative of the type of results which may be obtained on a small scale and used as a guide for metallurgical coke producers. The writers are of the definite opinion that more of our coke producers will be doing more blending of coals for metallurgical coke in the future than they ever have done in the past. Those plants that are equipped to blend coals from at least four different sources will be more fortunate than those with less blending capacity. Experimental work to determine the effect of each of the coals used in blending will result necessarily in considerable economy of operation. Undoubtedly, the easiest, cheapest, and quickest way to obtain the desired information on coal blends is by means of a small experimental coke oven in which the resulting physical and chemical properties of coke from a coal blend may be determined. This is the type of control work which may be pursued profitably by coke plants in any section of the country.

The Relation of Free-swelling Indexes to Other Characteristics of Some Alabama Domestic Stoker Coals

By Reynold Q. Shotts,* Member AIME

(Cincinnati Meeting, October 1947)

Introduction

The small domestic underfeed stoker as now designed is unusually sensitive to the coking and plastic properties of coals, and when the attempt is made to burn the high rank coking and caking coals of the Appalachian fields, serious trouble may occur. Part of the trouble may be caused by the ash content and clinkering characteristics of the particular coal in use, but more often it is the result of an excessive accumulation of coke in the form of "trees" or masses in the fuel bed. Often the zone of burning becomes so restricted that the fire goes out, and in extreme cases the fire may be lifted completely out of the fuel bed and away from the air ports. Even if the fire is preserved, excessive coking may cause response to subsequent heat demand to be slow and irregular. Coke trees seem to grow most readily during periods of hold-fire operation, so that in order to preserve the fire in mild weather it is often necessary to set the timer for such a long operating period each hour that much unneeded and unwanted heat is developed, causing discomfort. During demand periods the fire may pick up so slowly that the intense burning which develops just before stoker cut-off causes a serious overshooting of the thermostat.

Stoker coal performance tests which have been made in a number of laboratories[1,2,3,4,5] are one method of approach to the problem of selecting stoker coals. As yet these tests have not been standardized and they are expensive and time-consuming. The value of the results probably increases with the length of the test. Sherman and co-workers at Battelle Memorial Institute believe that even a one-week test is not too reliable a trial[2] and it is quite likely that others will agree. A great advantage of such tests is that characteristics of coals, other than the tendency to form coke masses, may be evaluated from them. The selection of stoker coals would be greatly facilitated if there existed some simple measure of the tendency of a coal to form coke in a fuel bed.

Although possessing all of the disadvantages of any small-scale laboratory test, the test for the free-swelling index of coal has been found useful for evaluating the probable effects of swelling properties upon combustion.[7] Ostborg, Limbacher and Sherman, have shown that the test can predict, roughly, the tendency of coke to accumulate in the fuel bed of a domestic stoker. They also gave instances of its value in predicting combustion behavior in some large installations. The use of the test for free-swelling index has become well established and has been adopted as a standard by the American Society for Testing Materials.[8]

When domestic stoker coal investigations began at the State Mine Experiment Station in the summer of 1944, it was decided

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[1] References are at the end of the paper.

to make free-swelling tests of all the coals investigated. It was further decided to attempt to relate the coking characteristics of the coals, as revealed in performance tests, to their free-swelling index. The necessary equipment, however, was not obtained and set up until the summer of 1945. By this time two series of stoker performance tests had been completed and preliminary accounts of them had been published.[9,10] The majority of the tests had been conducted with samples of coal comparatively fresh from the mine. When the third series of performance tests was made during the summer of 1945, swelling tests were made at the same time. The size of coal used for the preparation of the free-swelling test sample was the same as that used for the stoker test. The results of a complete group of eleven tests were thus made available for attempted relation to free-swelling indexes, even though they were made using stored coal. In September, 1945, grab samples of fresh slack weighing 50–100 lb each were taken at the same mines, with one exception, as those that had furnished the sample for the stoker coal performance tests of the first two series. One mine had been worked out and was closed in the interval between samplings, so that another mine on the same bed and about 10 miles away had to be sampled instead. A proximate analysis of a representative portion of each of the four-ton gross samples collected for performance tests is shown in Table 1.

TABLE 1—*A Proximate Analyses of Eleven Slack Coals Used in Stoker Performance Tests (Air Dry Basis)*

Coal No.	Moisture, Per Cent	Ash, Per Cent	Volatile Matter, Per Cent	Fixed Carbon, Per Cent	Sulphur, Per Cent	Heating Value, Btu per Lb
1	1.1	6.1	34.6	58.2	0.8	13980
2	1.5	6.5	35.5	56.5	0.9	14040
3	1.6	7.0	35.7	55.7	1.2	13690
4*	1.4	9.0	34.3	55.3	1.0	13540
5	1.1	10.0	35.3	53.6	0.9	13600
6	1.9	2.4	36.2	59.5	0.8	14540
7	2.6	3.9	36.4	57.1	1.5	13910
8	0.9	4.0	35.5	59.6	0.9	14230
9	3.0	4.1	37.2	55.7	1.3	13760
10	2.0	10.8	38.9	48.3	1.8	12910
11	2.7	3.6	38.4	55.3	1.4	13780

* Coal not washed.

FREE-SWELLING INDEXES

Table 2 gives the results of all free-swelling index determinations of the coals listed in Table 1. All eleven coals were used for two or more stoker coal performance tests. All of the samples, except that of Coal 4, represented washed coals. Fig 1 shows five representative groups of buttons having different swelling indexes and illustrating different shape characteristics.

TABLE 2—*Free-swelling Indexes of Eleven Fresh Washed Slack Coals from the Warrior and Cahaba Coal Fields of Alabama*

Coal No.	Bed	Size in Inches	Free-swelling Index
Cahaba Field:			
1	Clark & Gholson	−1½ + 0	8
2	Woodstock	−1½ + 0	6
3	Woodstock	−1½ + 0	6[4]
4	Woodstock	−1½ + 0	8[2]
5	Clark	−1¼ + 0	7½[3]
Warrior Field:			
6	Black Creek	−1¼ + 0	6½[3]
7	Black Creek	−1¼ + No. 30	3½[5]
8	Black Creek	−1¼ + 0	5½[3,6]
9	Black Creek	−1¼ + ½ mm	4½[3,4]
10*	Corona	−1½ + 0	4
11	Black Creek	−1¼ + ½ mm	3½[3,5]

* Not Mine No. 10 but another mine on the same bed located about ten miles from Mine 10.

[1] The sample was an arbitrary mixture of −⅞ in. + 0 slack and −4 in. + ⅞ in. nut. It was screened at 1½ in.

[2] This coal was not washed.

[3] The screen at the mine had 1½ in. round openings. The equivalent square opening is given to put all sizes on a comparable basis.

[4] When the fresh sample was taken some coal from stripping operation was being brought to the washing plant. The samples may, therefore, not represent completely unweathered material.

[5] In many recent U. S. Bureau of Mines publications the bed being worked is identified as the Jefferson, rather than as the Black Creek.

[6] Some stripping operations were shipping to the washing plant and about half of the coal was from another mine. All operations were on Black Creek bed.

Several distinctive characteristics of the coke buttons are noticeable. Actually only a few buttons conformed exactly to any one of the standard profiles. Variation in shape often was marked for individual buttons in the same series of four buttons.

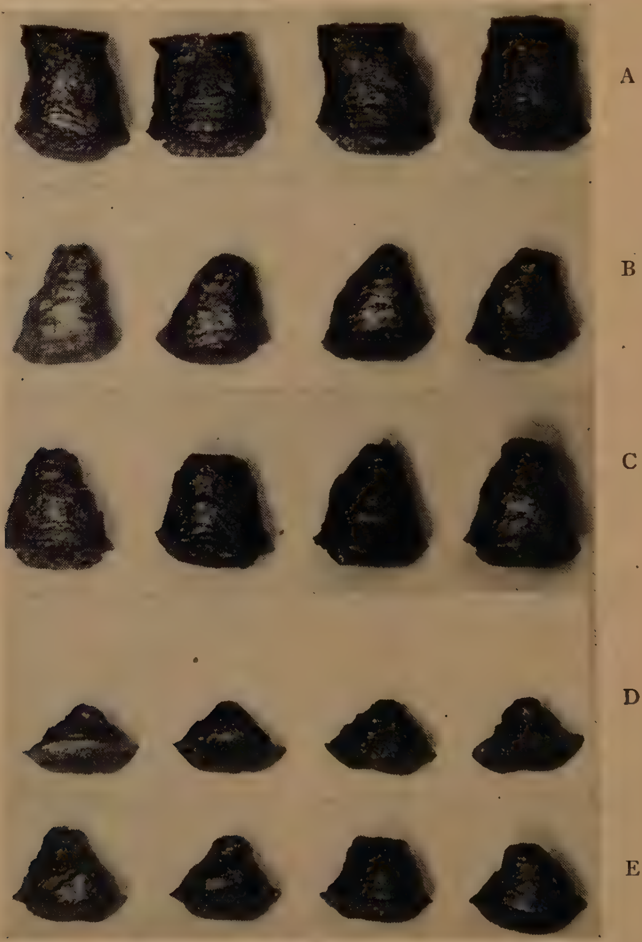

FIG 1—FREE-SWELLING TEST BUTTONS, FRESH WASHED SLACK.
(A) Coal 1. (B) Coal 3. (C) Coal 6. (D) Coal 7. (E) Coal 9.

In cases where the shape was such as to make it difficult to select the proper profile, an attempt was made to judge which profile would have fitted, given a properly shaped button with the same cross-sectional area. Subjective estimates of areas were made at the time, but since then the U. S. Bureau of Mines has suggested a quantitative method.[11]

The free-swelling test buttons that accompanied the third series of tests are not shown in Fig 1. In general these buttons, which were produced from slack which had been stored 6 to 14 months, were shaped like those made from the fresh coals but were seldom as symmetrical.

FREE-SWELLING INDEX AND GEOLOGIC STRUCTURE

Fig 2 is a map showing the areal distribution of free-swelling indexes for ten coals. There is some evidence of a relation between geologic structure and free-swelling index, although the map is by no means complete. Section *A-B* shows an increase in free-swelling index for the Black Creek group of beds from northwest to southeast. This is consistent with the increasing intensity of diastrophic disturbances of the beds. The major portion of the movement is attributed to Pennsylvanian or post-Pennsylvanian thrust forces from the southeast. Section *C-D* shows that mines on the Woodstock bed on the west side of the Cahaba field have free-swelling indexes of about 6, and that these increase to 7½ or 8 near the interior fault of that field. The mines near the fault on its east side also have high indexes. Both the Clark and the Gholson beds of the east side have been correlated with the Woodstock bed of the west side.[12] It would be of interest to learn whether a complete survey of the various beds above and below the Black Creek and Woodstock beds, in the Warrior and Cahaba fields respectively, would reveal similar relations between geologic structure and free-swelling index.

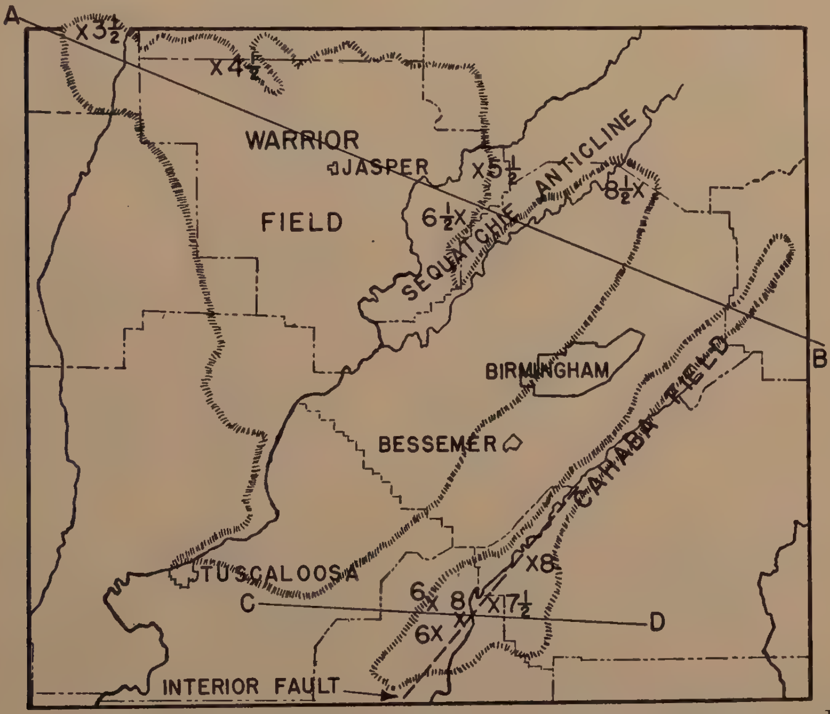

FIG 2—MAP SHOWING THE DISTRIBUTION OF SWELLING INDEXES FOR COALS FROM THE BLACK CREEK AND WOODSTOCK GROUPS OF BEDS.

FREE-SWELLING INDEX AND SIZE RANGE

Table 3 illustrates variation in free-swelling index with the size range of two washed slack coals. The sample of Coal 8 was a grab sample which had been mined only two weeks when the free-swelling indexes were determined. The sample of Coal 1 had been mined nearly three months at the time of making the tests, so that free-swelling indexes of fresh coal would almost certainly be greater than those shown for stored slack.

Coal 8 showed a small rise in free-swell-

TABLE 3—*The Variation of Swelling Index with Size for Three Alabama Coals*

Size	Coal 8[1]	Coal 1[2]
−1½ + ½ in.	5½	
−1½ in. + No. 4[3]	5½	
−¼ in. + No. 4		7
− No. 4 + No. 8		7
− No. 4 + No. 16	6	
− No. 8 + No. 16		7½
− No. 16 + No. 30		7
− No. 16 + No. 50	5½	
− No. 30 + No. 50		6
− No. 50 + No. 100		5½
− No. 50 + No. 200	3½	
− No. 100 + No. 200		5½
− No. 200	1	2½

[1] −1½ in. + 0 washed slack from Mine 8, Black Creek bed, Warrior field. Above ground less than two weeks when swelling tests were made.

[2] −¼ in. + 0 washed slack from Mine 1, Clark and Gholson beds, Cahaba field. Mined two or three months before free swelling indexes were determined.

[3] All sieve sizes refer to the U. S. Standard Scale.

ing index to a maximum of 6 at the − No. 4 + No. 16 size, with a steady decline in the successively smaller size intervals. The buttons from the − No. 200 material were so friable that they fell to a powder on handling. The material definitely was non-agglomerating.[6]

The maximum swelling index for Coal 1 occurred at approximately the same size interval as did that for Coal 8. The − No. 200 portion of Coal 1, however, was agglomerating. The buttons produced by the various size fractions of Coal 1 are shown in Fig 3.

The observed reduction of free-swelling index in successively smaller size fractions may have been caused by an increase in the quantity of inorganic matter, or to some differences in the composition of the coal substance itself. Unfortunately, neither ash analyses nor estimates of petrographic compositions are available for all the size fractions of either coal. Early work at the State Mine Experiment Station indicated that the presence of inorganic material had little effect upon free-swelling up to an ash content of about 20 pct. Later work has shown that beyond this ash content, inorganic matter lowers the free-swelling index noticeably for many coals. The non-swelling character of fusain and the swelling properties of vitrain have been demonstrated. It has been pointed out that there is a tendency for the former petrographic constituent to concentrate in the − No. 200 size and the latter in the sizes just larger and up to about No. 4.[13,14,15] Single, unchecked determinations of fusain at the State Mine Experiment Station, using the chemical method,[16] indicated 6 pct fusain for Coal 8, −1¼ in. + 0 slack and 49 pct fusain for the − No. 200 fraction. No determinations of fusain were made for Coal 1. The author believes variation in petrographic composition to be the principal cause for the differences shown for free-swelling index of the various size fractions of Coal 1 and Coal 8.

FREE-SWELLING INDEX AND TIME AND CONDITION OF STORAGE

The length of time that coal has been out of the ground and exposed to oxidation has a definite effect upon the free-swelling index. The effect is quite variable from coal to coal. It has been observed that any factor which reduces coking and caking, presumably by oxidation, acts more powerfully on weakly coking coals than on those with very strong coking properties.[6] This conclusion is confirmed by the results of oxidation in storage as shown in Table 4.

For every coal in Table 4, two and, in five cases, three conditions are available for comparison. All of the tests given in the column headed "fresh-slack" were made within two weeks after the coal was mined. It must be remembered, however, that in no case does the "fresh slack" sample represent the same lot of coal that is represented by the other two columns of swelling indexes. Both samples merely came from the same mine. The fact that the "fresh slack" samples do represent different lots of coal from those of the other sample, may serve to cast some doubt upon the comparability of the indexes. With all the results, however, uniformly indicating the same conclusion, it is unlikely that

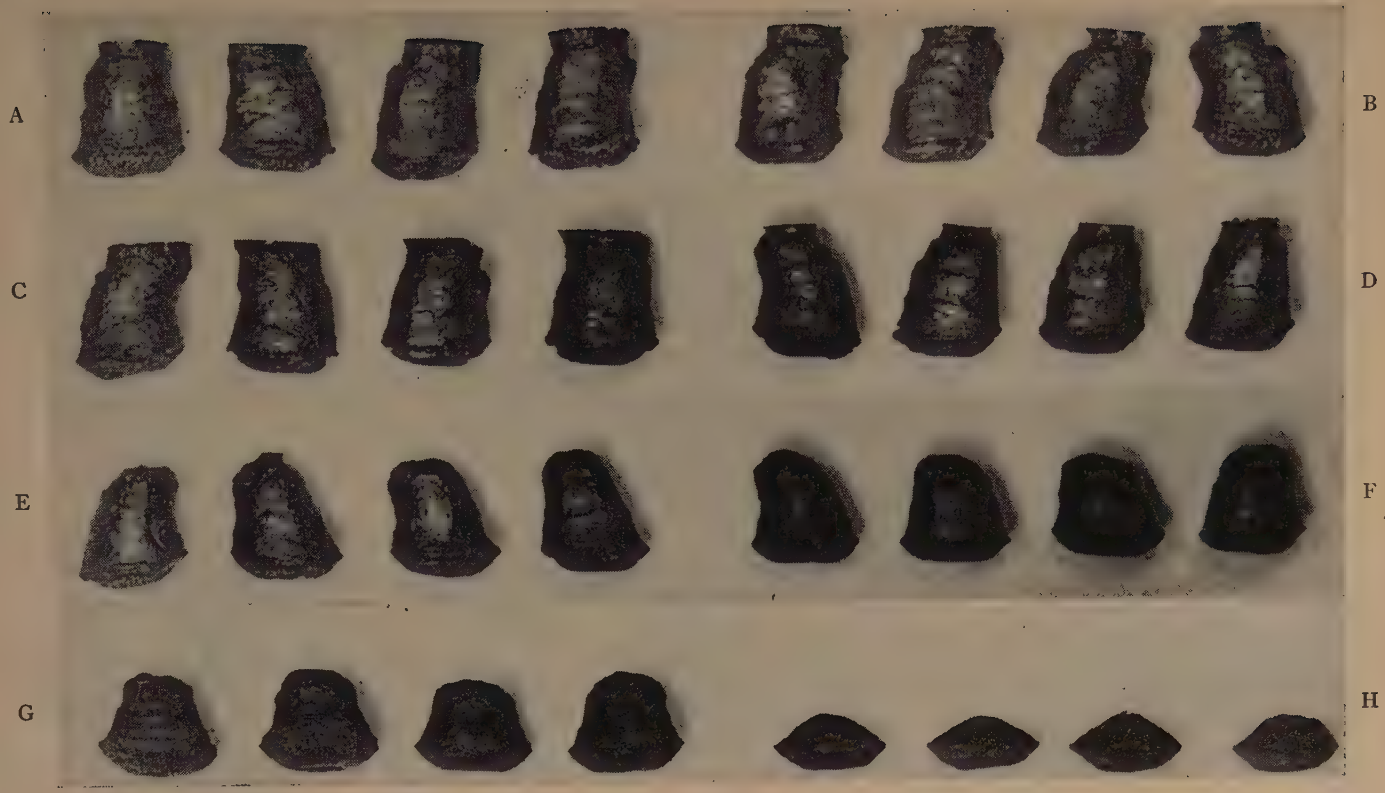

Fig 3—Free-swelling test buttons of Coal 1 stored slack for several size ranges.
(A) − ¼ in. + No. 4. (B) − No. 4 + No. 8. (C) − No. 8 + No. 16. (D) − No. 16 + No. 30. (E) − No. 30 + No. 50. (F) − No. 50 + No. 100. (G) − No. 100 + No. 200. (H) − No. 200.

samples from exactly the same lot would have indicated anything contrary to that shown.

The samples termed "stored slack" were the residue of the approximately 4-ton sample secured for stoker performance tests. They were stored inside in perfectly dry bins. Some of the samples were still wet when stored. After air drying, all of the coals from the Cahaba field contain less than 2 pct moisture and those from the Warrior field less than 3 pct. The number of days given for storage as slack are only approximate. The samples labeled "stored fines" were prepared from fresh slack and stored in rubber-stoppered 4-oz bottles in the analytical laboratory for approximately the number of days shown. The "days" column is only for time of storage after being reduced to pass a No. 60 sieve. The laboratory was quite warm—80 to 95°F—winter and summer, and the humidity was low, particularly in winter.

An examination of Table 4 reveals a reduction in swelling index with time of storage for every coal. Coals 4, 7 and 11 changed two units as the result of storage as slack.

In the cases of Coals 7 and 11, the indicated change was very great in proportion to the original low swelling index. Only Coal 2 showed a reduction of less than one unit.

The division of the coal into particles passing 250 microns increased the rapidity with which time of storage reduced swelling power. This change was to be expected because of the greater surface exposed to oxidation. Coal 9 showed the greatest reduction in swelling. The minimum reduction of swelling power that occurred from fresh coal was two units. Fig 4 shows the change in free-swelling index with length and condition of exposure for two coals.

It is obvious that by the time Coals 7, 9 and 11 had been stored as slack for a year they had become practically non-swelling. Unfortunately the relative change for the better was not so great for those coals that swell enough to cause possible trouble in the underfeed stoker. Storage under oxidizing conditions, or low cost methods for speeding up oxidation may present the possibility of converting unsatisfactory domestic stoker fuels into more satisfactory ones.

TABLE 4—*The Effect of Length of Storage Time and Particle Size of Stored Coal on Free-swelling Index*

Coal No.	Bed	Fresh Slack	Stored Slack	Approx. Length Storage Period, Days	Stored Fines Passing No. 60 Sieve	Approx. Length Storage Period, Days[2]
1	Clark & Gholson	8	6½	400		
2	Woodstock	6	5½	400		
3	Woodstock	6	5	350		
4	Woodstock	8	6	350		
5	Clark	7½	7	320		
6	Black Creek	6½	5½	300	4½	295
7	Black Creek[1]	3½	1½	270	1	260
8	Black Creek	5½	4	260	3½	105
9	Black Creek	4½	3	270	1½	260
10	Corona		5	220	4½	215
11	Black Creek[1]	3½	1½	240	1½	225

[1] In many recent U. S. Bureau of Mines publications, the bed being worked is identified as the Jefferson rather than as the Black Creek.

[2] Storage period given is that which elapsed between the time of reduction to pass No. 60 sieve and the determination of free swelling index. No coal had been stored as slack for more than one month at the time the — No. 60 sieve sample was prepared.

Free-swelling Index and Coke Accumulation during Performance Tests

One criterion which is frequently used as an indicator of the probable coking tendency of a stoker fuel is the weight of

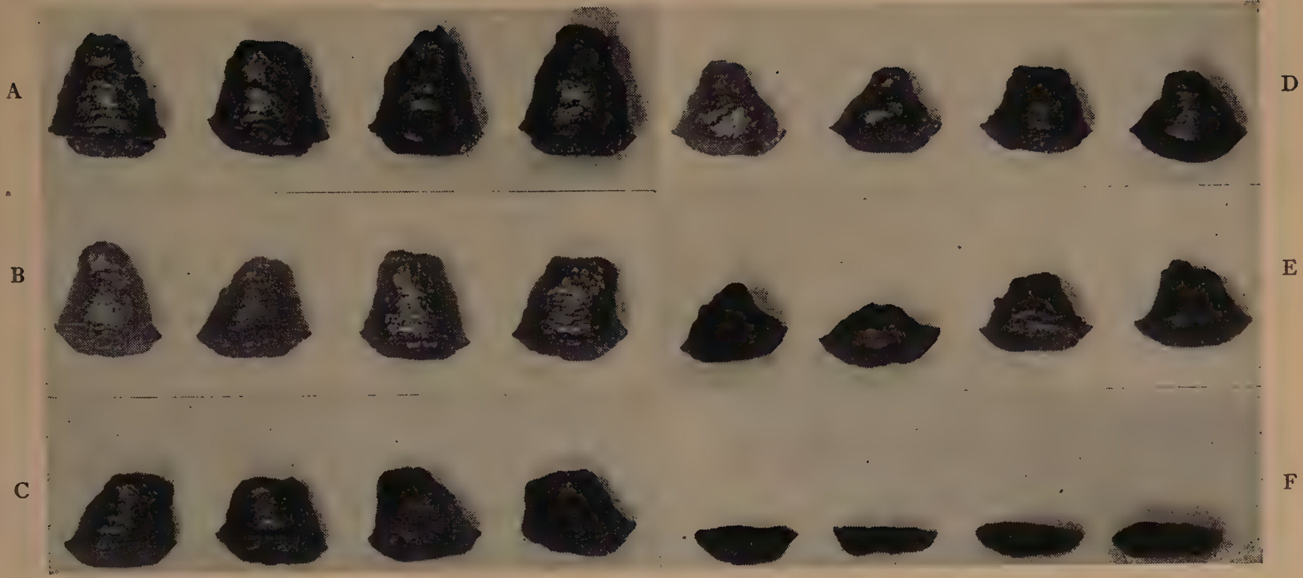

FIG 4—TEST BUTTONS SHOWING THE EFFECT OF STORAGE ON FREE-SWELLING.
(A) Freshly mined slack Coal 6. (B) Stored slack, Coal 6. (C) Stored Fines, Coal 6 (— No. 60). (D) Freshly mined slack, Coal 9. (E) Stored slack, Coal 9. (F) Stored fines, Coal 9 (— No. 60).

dry coke left on the fuel bed at the end of fixed-schedule stoker coal performance tests. For the tests of the first series made at the State Mine Experiment Station, coke was considered to be all of the combustible material retained on a ½-in. square opening screen. For tests of two other series, a 1-in. screen was used. The latter size limit was the one used by Ostborg, Limbacher and Sherman in their work on the relation of coke-forming tendencies to the free-swelling index of various coals.[7] As was pointed out by these investigators, the quantity of coke left on the bed is dependent upon the amount of excess air accompanying the combustion as well as upon the free-swelling tendencies of the coals. This finding was supported by those of Knox and Sawyer.[3] Small departures from any general relation may be explained by differences in air supply or by differences in its distribution to the fuel bed. Another factor that the author believes to be very important is the degree of care taken by the person removing the residual bed from the furnace to minimize variations in this hand operation from test to test. The friability of the coke and the way the coke

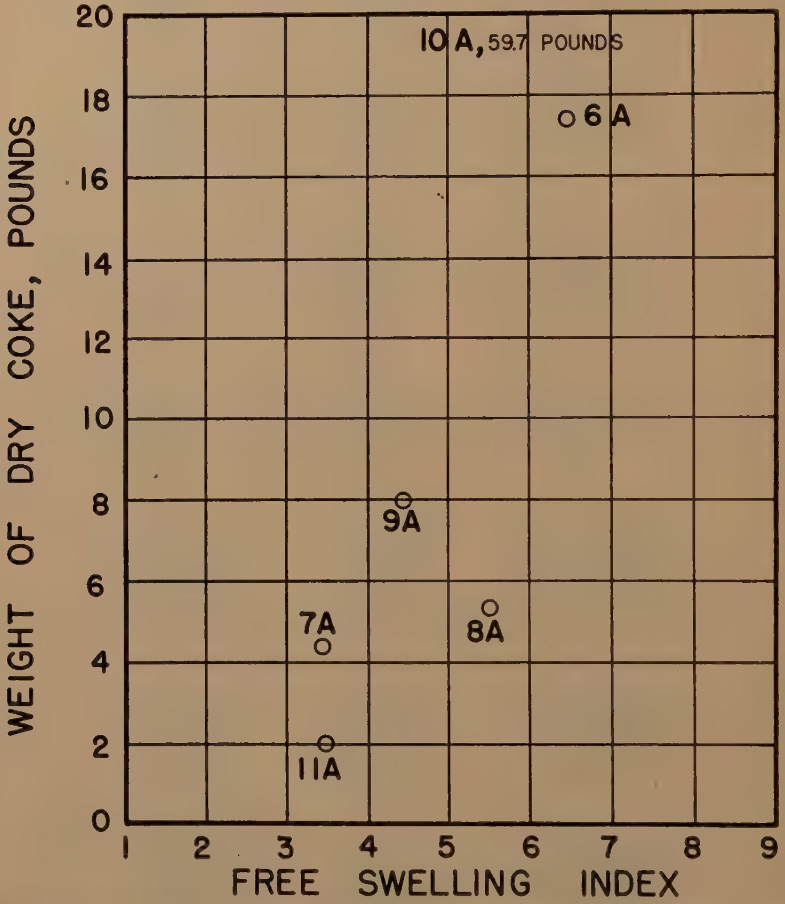

Fig 5—The Relation between coke accumulation and free-swelling index, long tests, second series.

and clinker are distributed in the bed will introduce variations in the determination of just what proportion of the coke present in the bed was really +1-in. material. having the lower free-swelling indexes, perhaps coke breakage will not throw suspected relations too far out of line.

Fig 5, 6 and 7 show the weight of dry, +1-in. coke left on the fuel bed at the end of each test, plotted as a function of the free-swelling index. For a few tests of the first two series, swelling indexes were estimated on the basis of the length of time the coal had been stored. This procedure is not very satisfactory but the estimate could hardly be in error by more than one unit.

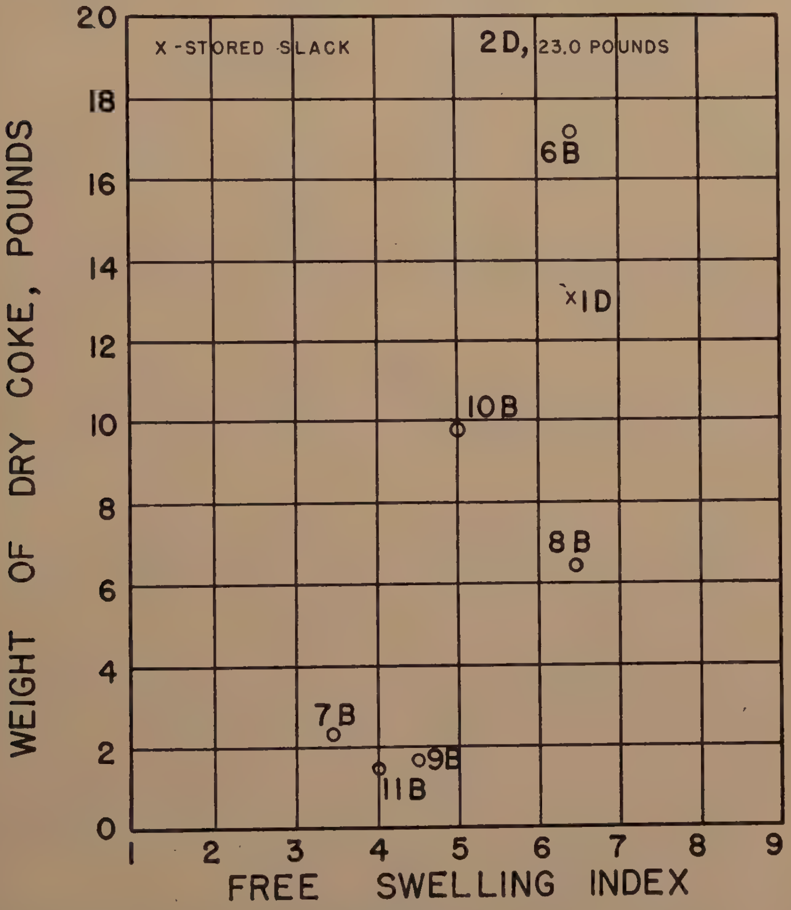

FIG 6—THE RELATION BETWEEN COKE ACCUMULATION AND FREE-SWELLING INDEX, SHORT TESTS SECOND SERIES.

Insofar as it was possible to do so, the same attendant was assigned the task of cleaning the furnace after each test, and an attempt was made to remove the material with a minimum of breakage. Even the most cursory observation will reveal that the cokes on fuel beds vary widely in friability. As the more friable cokes are likely to be the least objectionable, and since they are likely to come from the coals

The scattering of points is considerable, but there is an increase in +1-in. residual coke with increasing free-swelling index. All tests made with Coal 1 yielded somewhat less coke than the free-swelling indexes of the coal suggest they should yield. Coal 2 appeared to produce a larger coke residue than its swelling index indicated. This was particularly noticeable for fresh slack but occurred also for stored slack. The increase in the quantity of coke left on the fuel bed, with increase in free-swelling index, appeared to be quite regular up to an index of 5 or 6. Above this the rule of increase seemed to break down to some extent, and the free-swelling index ceased to be a certain criterion for the tendency to form coke masses. This apparent break at an index of 5 or 6 may have been caused by the choice of coals, or it may hold for all coals. Ostborg, Limbacher and Sherman say: "Experience in the laboratory has shown that coals which have a swelling index of 6, or less, will cause no difficulty from coke formations in the fuel bed of the small underfeed stoker. Those having indexes of 8 to 9 may or may not cause difficulty but are suspected." The index 6 appears a little high

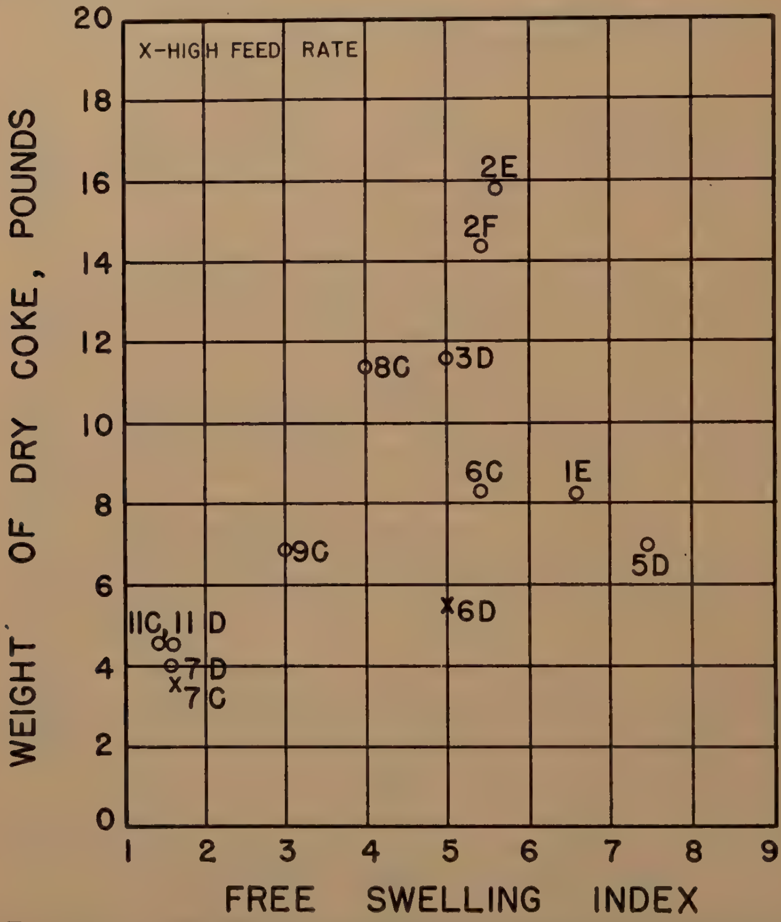

FIG 7—THE RELATION BETWEEN COKE ACCUMULATION AND FREE-SWELLING INDEX, TESTS, THIRD SERIES.

for some Alabama coals, but the observation that high indexes "may or may not" cause difficulty is confirmed to some extent by the erratic coke-formation results obtained in tests for coals of higher swelling index.

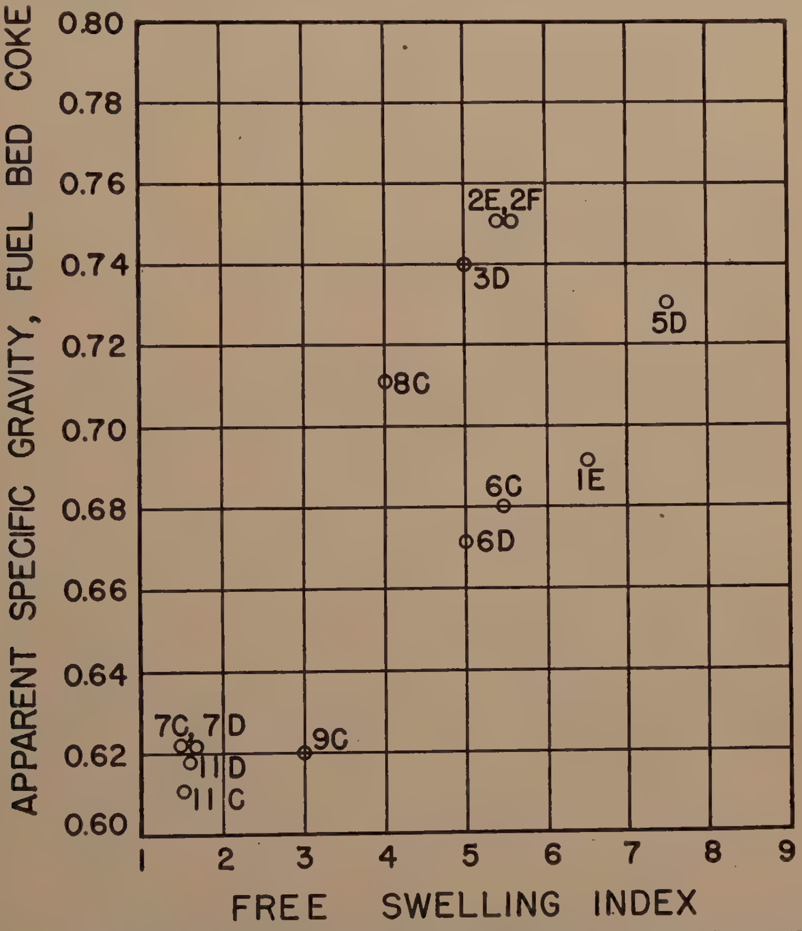

Fig 8—The relation of apparent specific gravity of fuel bed coke to free-swelling index.

Free-swelling Index and Apparent Specific Gravity of Coke

One other physical factor was examined in the tests of the third series, in the hope that it might contribute something in answer to the problem of excessive coke formation. The examination was an approach from the direction of the quality of the coke formed. The factor used was the apparent specific gravity of the coke. Presumably the higher the specific gravity the more difficult it is for air to penetrate a bed made up largely of coke and the more the reaction surface between the coke and air is restricted, the more slowly the coke burns.

The apparent specific gravity of fuel bed coke was determined by a modification of the method outlined in ASTM Designation: D 167-24 for the determination of the volume of cell space of lump coke.[17] The method is intended for use with byproduct and beehive coke and may not be suitable for the material found on the domestic stoker fuel bed. However, by the use of the ASTM method the results, even if erroneous in any absolute sense, should be comparable.

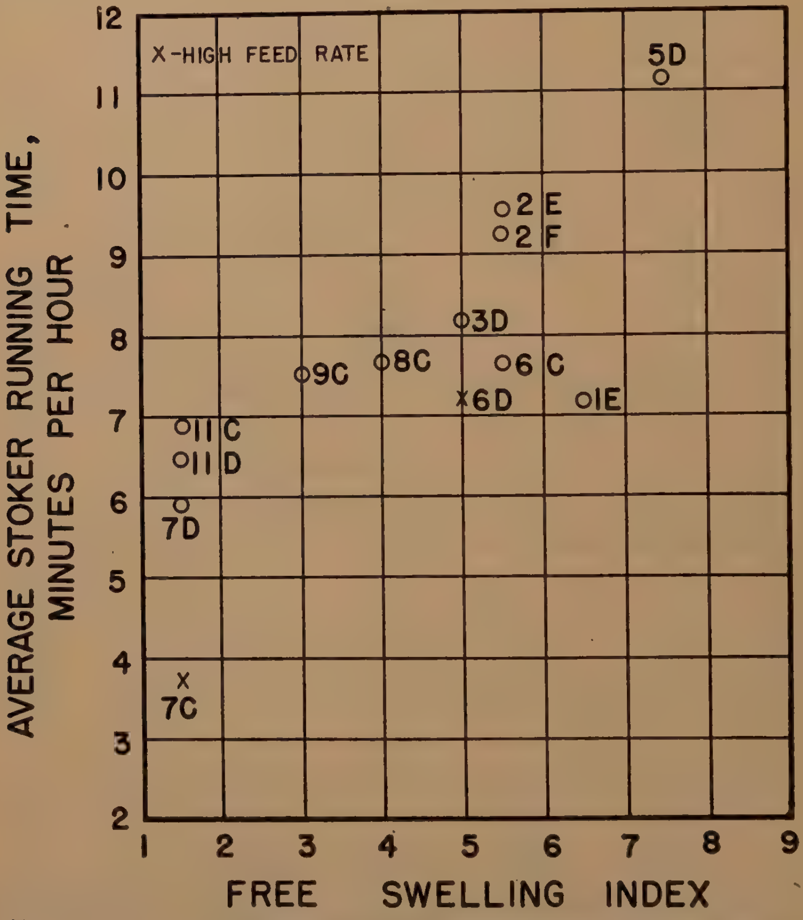

Fig 9—The relation of the average length of stoker running periods during hold-fire operation to free-swelling index.

Fig 8 shows the relation between the apparent specific gravity of coke and the free-swelling index. The distribution of points is quite similar to that for the other relations that have been plotted. Test 1E maintains its favorably low position. Certainly in a very general way the apparent specific gravity of the coke found in the fuel bed is proportional to the free-swelling index. Three duplicate tests on the same coal were in excellent agreement and indicate that the method yields reproducible results. Tests 6C and 6D gave results that were close, but the difference in size range of the coal feed made it unlikely that they would form cokes with identical physical properties. The explanation for the failure of Tests 1E, 5D, 6C and 6D to leave a quantity of coke on the fuel bed proportional to the free-swelling indexes of the coals, may lie partly in the comparatively low densities of their residual cokes.

Free-swelling Index and Hold-fire Ability

The criterion used in the third series of tests for judging the ability of a coal to hold fire was the relative average number of minutes running time required to bring the fire up to a predetermined condition once each hour. The relation of the average running time to the free-swelling index of the coal is shown in Fig 9.

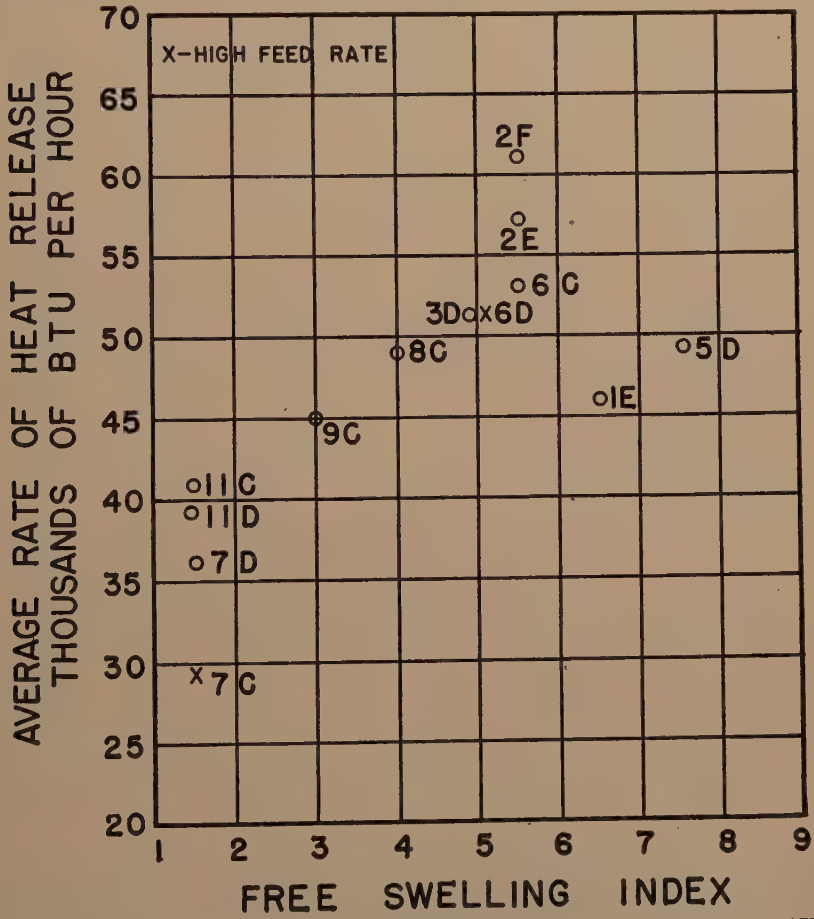

FIG 10—THE RELATION OF AVERAGE RATE OF HEAT RELEASE DURING HOLD-FIRE OPERATION TO FREE-SWELLING INDEX.

The distribution of points in Fig 9 is quite similar to that of Fig 7. Test 5D is an exception with a long average running period corresponding to its high free-swelling index. The very low running time of Test 7C is explained by the fact that a much faster feed rate was used with that test than with the others (except Test 6D). Tests 2E and 2F, along with Test 5D, gave average running times somewhat above the general trend, while the results of Test 1E were much below the trend.

Not only should a stoker coal be able to hold a fire during periods of mild weather, but the quantity of heat given off while doing so should be low. If this condition

TABLE 5—*Results of a Statistical Study of the Length of Running Periods during Hold-fire Operation*

Test No.	1E	2E	2F	3D	5D	6C	6D	7C	7D	8C	9C	11C	11D
Mean running time, min. (x)	7.6	9.6	9.6	8.5	11.6	7.4	7.7	4.1	6.3	7.7	7.6	6.9	7.1
Length of max. period, min	26	29	25	18	19	27	15	6	9	11	14	11	11
Standard deviation, min. (σ)	6.2	6.3	6.5	4.6	4.9	6.5	3.3	1.1	1.5	2.0	2.8	1.7	1.8
Coefficient of variation, pct (v)	81	65	68	55	42	88	44	26	24	25	36	25	25

Note: No. of cases (n) in each test = 30.

does not exist the stoker operates uneconomically and the house may receive unneeded and unwanted heat. Fig 10 shows free-swelling index plotted against rate of heat release in thousands of Btu per hour, during the hold-fire period of the tests of the third series. The distribution of points in Fig 10 again is quite similar to that of Fig 7. The very short running period in Test 7C accounts for its very low heat release rate. Tests 2E and 2F again gave results higher than the general trend, and Test 1E had a favorably low heat release rate for a strongly swelling coal. The low heat release rate of Test 5D, contrasted with its long average running period, suggested a very slow "pick-up" rate for that coal.

In winter weather of the type that often occurs in the South, cold frosty mornings require a rapid pick-up of the fire but as the temperature rises during the day it is desirable that no thick, rapidly burning fuel bed be present to give off unwanted heat. For this type of operation, a coal is needed which will respond rapidly and steadily to changes in heat demand. In an effort to find some criterion for measuring the factor of steadiness of response of various coals, a statistical study was made of the length of running periods that occurred during hold-fire operation in the third series of tests. The length of each individual period could be estimated from the flue gas temperature recorder chart to the nearest one minute. The results of the calculations are shown in Table 5. The method of calculation followed that recommended by the American Society for Testing Materials in the "Manual on Presentation of Data," April 1943 reprinting, p. 13–21.

Apparently the same coal in essentially duplicate tests tended not only to show practically the same mean running time for each test, but the variations about that mean—the swings above and below it—appeared to form the same pattern. In the case of the two tests with Coal 7, difference in air supply and feeding rate resulted in considerable difference between the two means and the standard deviations. When the latter, however, was expressed as a percentage of the former—the coefficient of variation—the figures were almost identical. They were identical for the two tests with Coal 11. Since Coals 7 and 11 are mined from the same bed and within sight of each other, four very close checks were obtained for essentially the same coal, despite variations in feeding rate, burning rate, and air supply. Tests 2E and 2F yielded close agreement for coefficient of variation in almost duplicate tests for a coal of a very different type. The great difference between the coefficient of variation for Tests 6C and 6D brings up the question of the effects of the removal of the finer sizes upon stoker coal performance. The lessened blocking of air by coke formation, when the finer sizes were absent, probably resulted in better distribution of air to the bed and consequently in a much more uniform response for Test 6D. The similar values for the coefficient of variation for Tests 5D and 6D both of which had feeds of large size consist tend to support this conclusion.

Fig 11 shows coefficient of variation plotted against free-swelling index. It indicates that the more strongly swelling coals are more erratic in their response than are those with lower indexes. Tests 5D, 6C, 6D and 8C are somewhat out of line. No explanation is apparent for the relatively large coefficient of Test 6C or for the relatively low one of Test 8C, but unquestionably Tests 5D and 6D benefited from the large size consist of coal used in them.

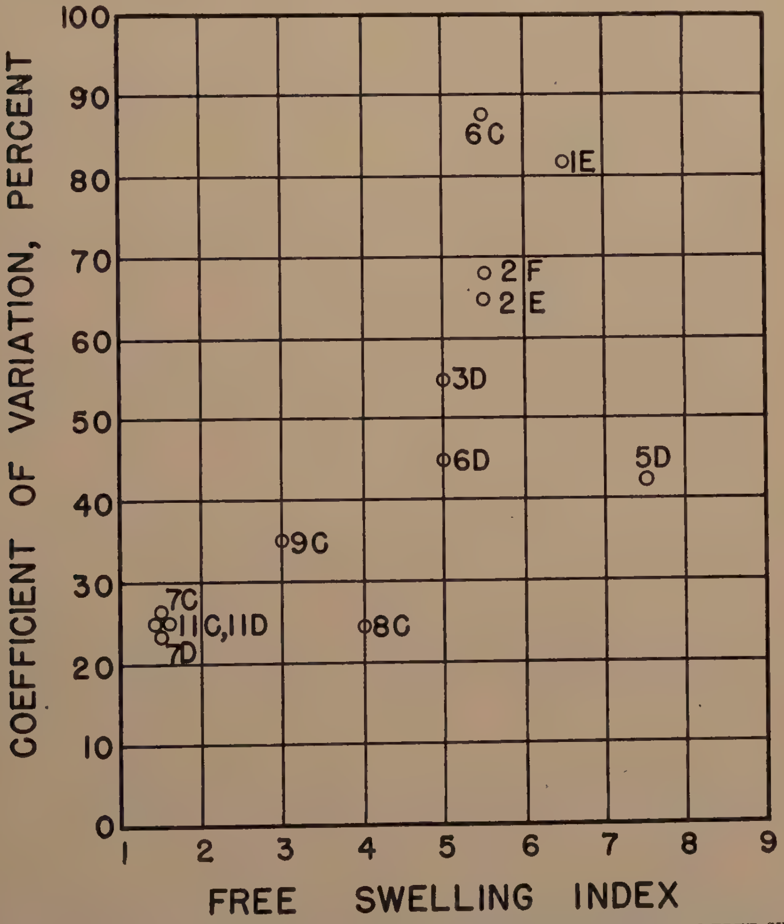

FIG 11—THE RELATION OF COEFFICIENT OF VARIATION OF STOKER RUNNING TIME TO FREE-SWELLING INDEX.

When the conventional program timer with its fixed-length and fixed-interval running period is used to provide hold-fire control, it is reasonable to expect that coals showing a wide variation in running time from period to period in tests of the type described would be the more unsatisfactory with respect to hold-fire characteristics. With the conventional type of timer control the setting must be set high for all cycles, so that the stoker will run long enough to build up the fire satisfactorily when a cycle requiring a long running period occurs. If the stoker does not run long enough during such a cycle the fire may get so low that it will go out before the next running period. If, however, the timer is set high enough to build the fire to a satisfactory level for such cycles much extra coal will be fed in and some useless burning will take place during stoker-idle periods as well as during those running periods in which prolonged operation is unnecessary. Under such conditions fuel economy is greatly reduced and unwanted heat is released. If a coal exhibits uniformity in the running time required to build up the fire (low coefficient of variation), the timer may be safely set near some average time deemed the minimum necessary for the preservation of the fire, with little fear that extraordinary cycles will occur during which the fire will go out. In the case of a coal with a high coefficient of variation in running time a setting far above any average time thought necessary for the preservation of the fire will be necessary in order to prevent out fires during the cycles in which the fire becomes unusually low. The selection of the proper minimum running time in any case will be a trial and error procedure because all the averages obtained in tests of the type described here are too high for economical use in an actual installation. In the tests relative values only were of interest.

From the available data, the discussion of it in the paragraph above, and from Fig 11, three conclusions seem justified: (1) Coals having a high coefficient of variation in running time for hold-fire tests of the type described probably will require a higher setting of the conventional timer for fire preservation than will those with a low coefficient of variation. (2) For such coals, a hold-fire control actuated by flue gas or fire box temperature probably will be much more satisfactory than the conventional timer. (3) Coefficient of variation of running time seems to be related to free-swelling tendency, but the relationship is not exact. Variation in coefficient is particularly wide for coals with indexes of 4 to 6.

Free-swelling Index and Unit Coal Heating Value

The resemblance between all of the plots of test results against free-swelling indexes suggest that the free-swelling index is a semi-quantitative attempt to measure some fundamental characteristic of the coal. That the characteristic is fundamental is shown by Fig 12 and 13, in which the free-swelling index for each coal is plotted against its dry, mineral-matter-free, heating value. Errors and difficulties of analysis and the semi-quantitative method necessary for the determination of free-swelling index result in a considerable scattering of points and some overlapping, but in spite of this the free-swelling index is a fairly good indicator of the rank of the coal insofar as rank is determined by heating value. All the coals tested were of bituminous A rank according to the ASTM classification[17] and it is quite doubtful if the indicated heating value-swelling index relation would hold for coals of other ranks, particularly higher ones. The very fact, however, that oxidation during storage not only lowers heating value but also affects the free-swelling and other rheological properties illustrates a close relationship. As far as Alabama bituminous A coals are concerned, it appears that the dry, mineral-matter-free heating value is as good an indication of probable stoker coal performance, from the standpoint of coke accumulation, as is the free-swelling index.

As a means of measuring the effect of free-swelling characteristics upon stoker coal performance, the free-swelling index was, in this study, only moderately successful. The reasons for its failure to classify some of the coals in a manner consistent with the way the others were classified are not entirely evident from the test results. Some operating conditions that were not properly evaluated or that possibly were overlooked completely may account for the results; or they may have been the consequence of some physical or chemical properties peculiar to these coals. The test conditions of air supply, feed rates and drafts, were not kept strictly constant, but attempts have been made to allow for variations in these factors in the interpretation of test results. Further study will be necessary to provide a complete explanation.

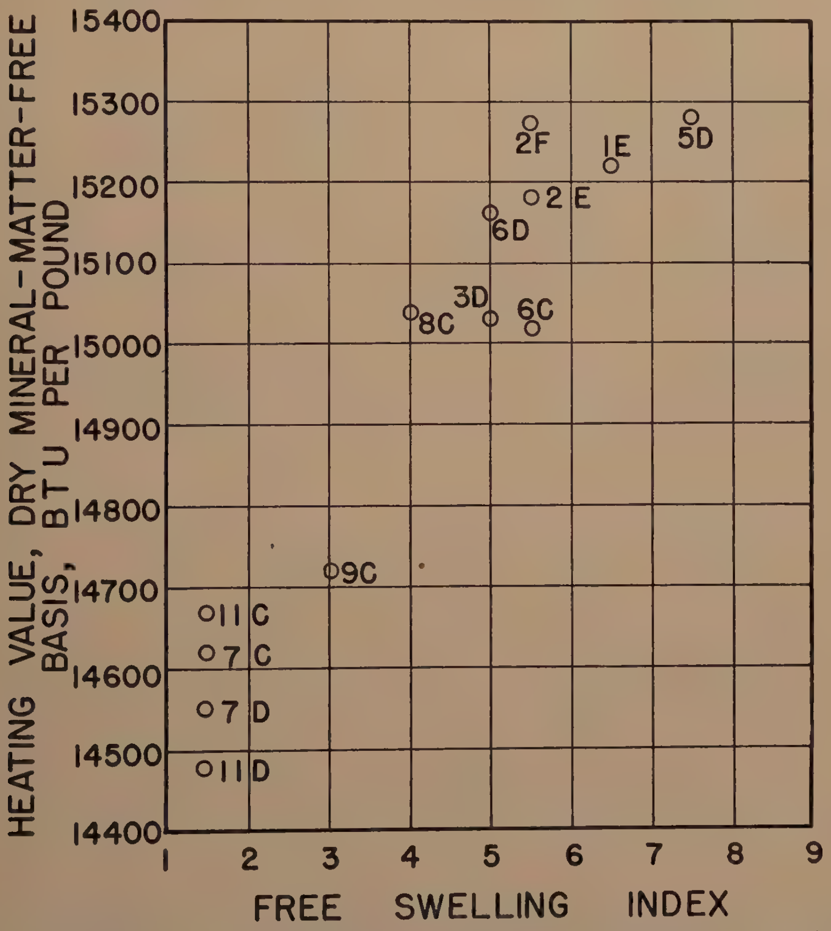

Fig 12—Free-swelling index as a function of dry, mineral-matter-free heating value, third series tests.

Summary

1. Free-swelling indexes of eleven Alabama bituminous coals which were used for stoker coal performance tests were found to range from 3½ to 8 units.

2. For two of the coals the free-swelling index decreased steadily with successively smaller size intervals, for sizes smaller than a No. 8 or No. 16 sieve.

3. Exposure to oxidation decreased the free-swelling index for all coals tested, with the greater relative decrease being shown by the coals with the initially lower indexes. Division into particles passing 250 microns accelerated the loss of swelling properties.

4. The work of other investigators indicating that the quantity of large coke left on the fuel bed after a fixed schedule test

run is roughly proportional to the free-swelling index of the coal feed, has been confirmed for Alabama coals.

5. The relation between fuel bed coke accumulation and free-swelling index holds best for Alabama coals with swelling indexes of five or below. Coals having indexes of five or above are more irregular in their tendency to form coke, a finding also mentioned by other investigators.

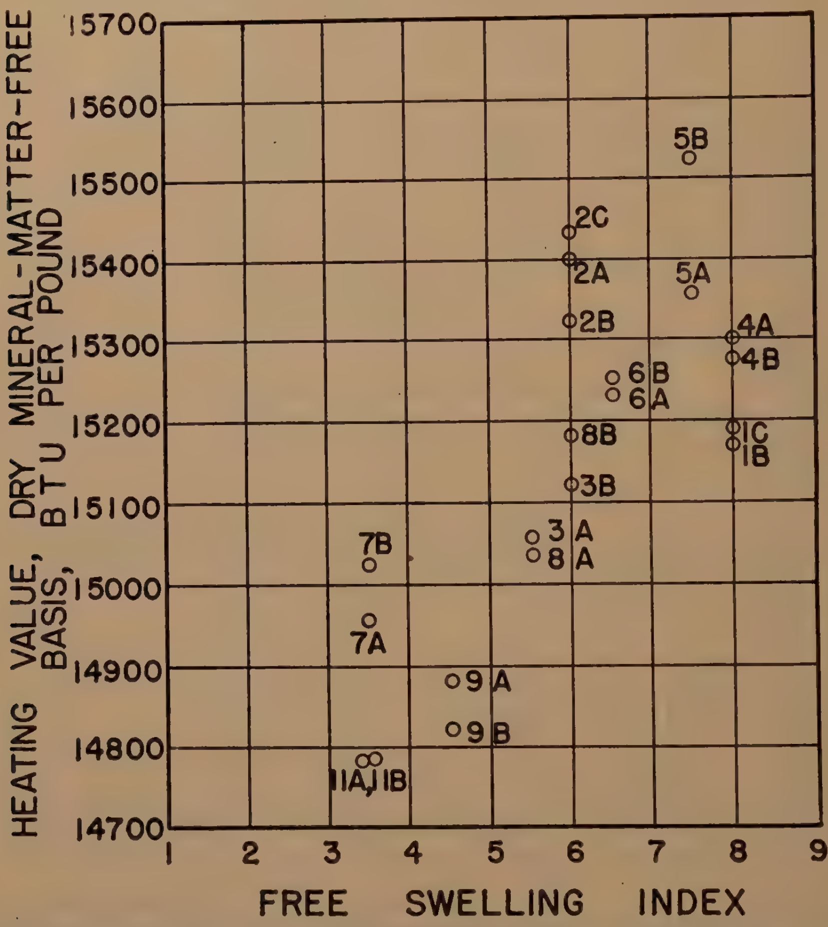

FIG 13—FREE-SWELLING INDEX AS A FUNCTION OF DRY, MINERAL-MATTER-FREE HEATING VALUE, FIRST AND SECOND SERIES TESTS.

6. The apparent specific gravity of fuel bed coke is roughly proportional to the free-swelling index of the coal feed. Variations in the relationship are fairly wide. Coals which left small quantities of fuel bed coke in proportion to their free-swelling indexes also showed a relatively low apparent specific gravity for their coke.

7. The average length of running period necessary to preserve a predetermined fire condition during hold-fire operation, the average rate of heat release during hold-fire operation, and the heat content of the coal on a dry, mineral-matter-free basis correspond quite well with the free-swelling index of the coal feed. In general, the individual cases which departed most widely from the general trend in the rela-

tionships mentioned here were the same ones which departed most widely from the average trend in the other comparisons.

8. The coefficient of variation of the lengths of the individual stoker operation periods corresponded in a general way to the free-swelling index. The relation was better when it was restricted to stoker feeds of approximately the same size composition. The magnitude of the coefficient of variation indicates the relative ease or difficulty of fire recovery from period to period and may be significant as a measure of the hold-fire ability of the coal. Tests in which closely sized feeds were used gave relatively low coefficient of variation even for strongly coking coals. The increased uniformity of fire recovery during stoker operation periods may account for the improved hold-fire ability which is often noted when the fines are removed from a stoker coal.

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DISCUSSION

(*William G. Christy aud R. A. Sherman presiding*)

J. D. CLENDENIN*—This is an interesting contribution by Professor Shotts, but it is unfortunate that certain of his observations were not investigated further. The free-swelling behavior that was noted for certain size fractions and partially oxidized samples is suggestive of behavior noted by others. For example, Mott and Wheeler[18] have pointed out that the free-swelling indexes of the bright and durain components of some British coals are not additive. The results described by McCabe, Konzo and Rees[19] for hand-picked durain, clarain and vitrain for an Illinois coal indicate nonadditive behavior for these components. These three components were assigned free-swelling indexes of 1, 3 and 5, respectively, yet these investigators noted indexes of 3 each for durain- and clarain-rich samples and 4½ for a vitrain-rich sample, all presumably from the same coal. In addition, the free-swelling behavior of the bright and dull-coal portions of a Lower Kittanning seam (ca. 36 pct V.M.) coal tested in our laboratory have been observed as nonadditive; the whole coal had a free-swelling index of 8, the bright-coal portion an index of 8½ and the dull-coal portion an index of 2.

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* The Pennsylvania State College.

[18] The Quality of Coke. 360–368 (1939) (London). Chapman and Hall.

[19] Ill. Geol. Survey. *R. I.* No. 78 (1942).

On the other hand, Barraclough[20] observed behavior in the crucible swelling test that appeared to be essentially additive when binary mixtures of an oxidized and an unoxidized coal, each from the same seam, were tested. How binary mixtures of other coals, as contrasted with coal components, would behave is a matter for investigation, since it is known that binary mixtures of coals when subjected to different conditions of heating do not necessarily behave additively.

The relation of free-swelling index to unit coal heating value may be deceptive; moreover, one would hardly choose the latter for any preliminary evaluation. British fuel technologists have long given attention to the relation of such properties as swelling index to unit coal properties, as carbon, hydrogen and heating value. Apparently, the correlation of swelling number with carbon and hydrogen has proved most suitable, but is of only limited value at best.[21]

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[20] The Pa. State Coll., B. Sc. Thesis, June, 1941.

[21] *Coke.* (Sept. 1943) **5**, 165–170.

Low Temperature Reactions of Oxygen on Bituminous Coal

By H. C. Howard*

(New York Meeting, March 1947)

Reaction of oxygen and bituminous coal starts as soon as the coal bed is exposed to air and, with some coals, proceeds with significant velocity even at normal temperatures and at normal oxygen partial pressures of one-fifth of an atmosphere. At higher temperatures, the reaction is greatly accelerated and ultimately a condition is reached where the process is very rapid and we then speak of combustion. It is the purpose of the present paper to discuss the reactions of oxygen and coal at temperatures below those of active combustion. It is these low temperature oxidation reactions which are responsible for heating in storage and for marked changes in agglutinating properties. These reactions may also be employed for the preparation of organic acids, which are of direct interest to the chemical industry.

For convenience in discussion, the low temperature reactions of oxygen and bituminous coal can be roughly divided into three stages though there is, of course, overlapping. Stage 1 is a surface oxidation stage, characterized by the addition of oxygen, with the formation of oxygen containing groups some of which have acidic properties. Carbon dioxide, carbon monoxide and water may be formed concomitantly and are always liberated at higher temperatures, and in cases where the oxidation is prolonged. Superficially, the coal is little affected by this primary stage of oxidation and usually only minor changes in ultimate composition result. There may be definite changes in coking and coking properties, and heat of combustion susceptibility to further oxidation. The presence of acidic groups can be demonstrated by specific reactions. These groups may show cation exchange properties like the mineral zeolites. The formation of oxalic acid has been reported. If the oxidation is continued, a large fraction of the organic material of the coal is converted ultimately into alkali soluble, acid insoluble products, which have been given the name "regenerated humic acids" and which are the characteristic products of Stage 2. These humic acids, in the solid stage, do not differ greatly in appearance from the original coal, but show large differences in chemical composition and chemical and physical properties. Along with the humic acids, carbon dioxide and some of the water soluble types of Stage 3, are also formed. If the oxidation is carried still further, the humic acids may be degraded completely to the simpler, lower molecular weight acids of Stage 3 which are characterized by solubility in acid as well as in alkaline media, and by their orange, yellow or white colors. Additional carbon dioxide is formed in the transformation from Stage 2 to Stage 3 and, for a number of American coals, a maximum yield of the water soluble types of acids is reached when about 50 pct of the carbon of the coal is converted to carbon dioxide.

Primary Oxidation

The initial step in the reaction of gaseous oxygen on coal, like that on carbon,

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appears to be one of addition with the formation of a surface complex, followed by decomposition of this complex and liberation of oxides of carbon and of water.

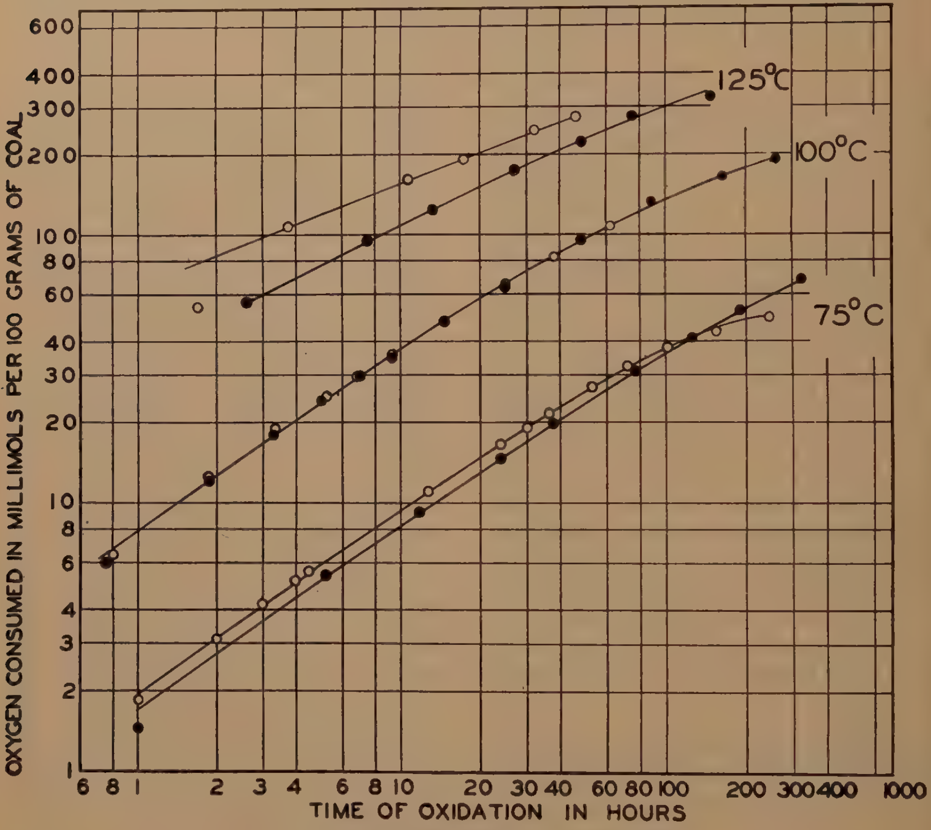

Fig 1—Rate of oxidation of Illinois coal.

Studies have been made of the rates of these primary reactions of oxygen with various bituminous coals, under a variety of experimental conditions, and a number of interesting relations have been established. As would be expected the rates of the primary reactions are a function of the rank of the coal, the surface area, the temperature, oxygen partial pressure and the time that the oxidation has proceeded.

Data have been reported[1] for nine bituminous coals, ranging in volatile matter from 17.2 to 38.2 pct, which show an approximately three-fold change in specific rate of oxidation over this range of volatile matter. Determinations on 68 bituminous coals, in the Coal Research Laboratory,[2] of the temperatures at which a coal reaches a constant reaction rate, have shown relatively small differences over a range of volatile matter from 18 to 32 pct. Outside of these ranges of volatile matter, very marked differences are noted.

Very little quantitative information is available on the relation between rates and surface area because of the uncertainties in measurement of the latter quantity. Rates of oxidation have been measured for samples of a British coal[3] screened into four size ranges; 2 to 10,

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[1] References are at the end of the paper.

10 to 30, 30 to 60 and minus 200-mesh. The areas of these samples, calculated by the Needham and Hill[4] relation, were 0.55, 7.8, 24 and 314 sq m-kg. The characteristic rates of oxidation of these samples were 0.00718, 0.0217, 0.0336, and 0.0514 g of oxygen per 100 g of coal per day. Thus for an approximately 600-fold increase in calculated area only an approximately ten-fold increase in rate was observed. Schmidt[5] has suggested, as a first approximation, that the rate is proportional to the cube root of the calculated surface. The much higher than anticipated rate of reaction of the coarse coal may result from considerable reactive, internal surface as well as from the presence in the larger mesh sizes of significant amounts of fine coal as a surface coating.

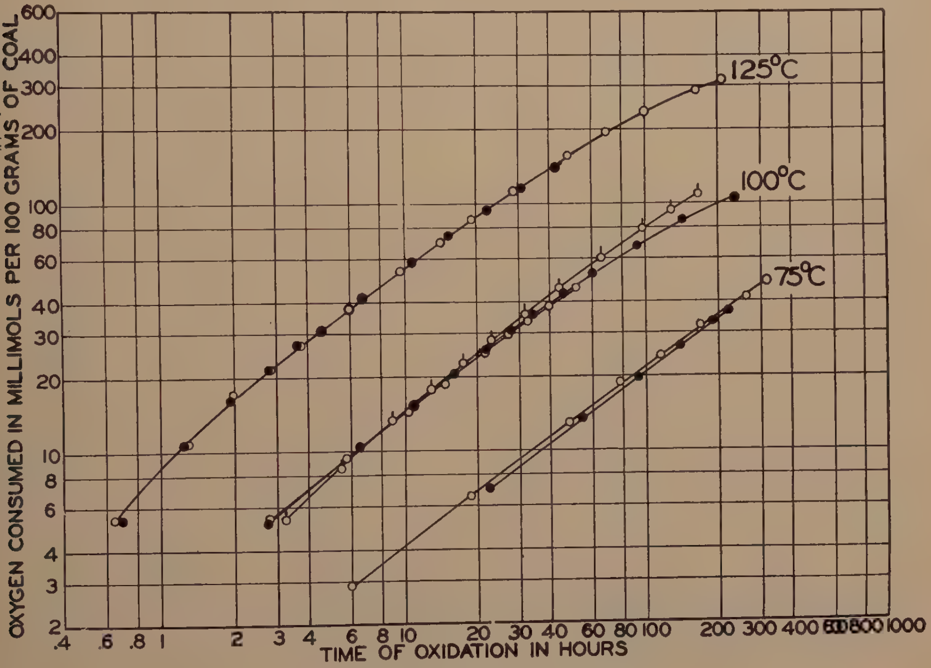

Fig 2—Rate of oxidation of Pittsburgh coal.

More data are available on the relations between rate and reaction temperature. In general, the available data fit an Arrhenius type equation reasonably well and the expected linear relation between the log of the rate and reciprocal of the absolute temperature is obtained at least over a limited temperature range. The so-called heat of activation, which is regarded as a measure of the difficulty of bringing about a chemical reaction, can be calculated from the slopes of these lines. Values so calculated from the low temperature reactions of oxygen on several coals are shown in Table 1. Even the highest values found are lower than those usual for chemical reactions where primary bonds are broken.

The effect of oxygen partial pressure on rate has been investigated by a number of workers and the data are summarized in Table 2. Calculations, based on the Coal Research Laboratory reactivity index test, of the effect of oxygen partial pressure on reaction rate indicate a value of "a" of 0.7 to 0.8 over the temperature range 200 to 250°C. Thus all the experimental data available show that the rate is proportional to less than the first power of the oxygen consideration. There is some indication that the value of the exponent increases with increase in temperature.

TABLE 1—*Effect of Temperature on Rate*

Coal	Heat of Activation	
	30–90°C	90–140°C
Barnsley[6]		
Softs	16,600	25,200
Top Softs	17,100	24,400
Hards	16,500	28,600
Jacks	15,600	25,900
Cannel	14,000	25,900
	75–125°C	
Pocahontas[7] No. 3	13,900	
Pittsburgh[7]	14,000	
Illinois No. 6[7]	13,600	

One of the most interesting kinetic characteristics of the low temperature reaction of oxygen on coal is the very marked decrease in rate of oxidation with time. Schmidt and Elder have recently shown[9] that this relation is satisfactorily represented by the following expression: $x = ct^b$ when x is percentage oxygen absorbed, c is a constant, t is the time and b is another constant. A log-log plot of the quantities x and t gives lines of slope b and intercept c. Representative data are shown in Fig 1, 2 and 3. A very interesting fact brought out by these plots is that the slopes of the lines are nearly constant for all the coals studied over the entire temperature range, that is, the values of b are nearly constant. The values of the intercepts c are, however, a function of both temperature and rank of coal. It is evident that the reaction is hindered by the accumulation of oxidation products on the surface and that this effect is relatively the same for all the coals studied and over a considerable temperature range.

The development of analytical methods for determining the exact nature and extent of this surface reaction has been a baffling problem. If the process consisted solely in the addition of oxygen, even in small amounts, methods for determining the extent of the reaction could, no doubt, be worked out. Since, however, significant amounts of the oxidation products are volatile and the coal is physically heterogeneous, very considerable changes in physical properties, without marked changes in ultimate composition, may appear, caused by the selective removal of specific parts of the structure in the form of volatile oxidation products. Data from a number of sources show that the low temperature reactions of oxygen on coal result, along with the addition of oxygen, in the removal of much larger amounts of hydrogen, on an atomic basis, than of carbon. The atomic carbon-hydrogen ratios in the volatile oxidation products are very much lower than in the original coal. This is illustrated by the data of Table 3. Since it is the hydrogen-rich parts of the coal structure, which are probably responsible for the greater part of the plasticizing action in the coke forming process, it is easy to understand the effects of small amounts of oxidation in decreasing the coking and swelling properties of coals. In this connection it is of interest to observe that both elemental sulphur and chlorine, which are chemical dehydrogenating agents, also react on coal to form products of greatly reduced agglutinating properties.

TABLE 2—*Effect of Oxygen Concentration on Rate*

Rate = $k[O_2]^a$

Coa	Deg C	Pct O_2	Values of a
Barnsley[8]			
Hards	30	5–100	0.5
Pittsburgh[9]	99.3	3–14	0.590
Lower Banner[9]	99.3	0.2–15	0.596
Pocahontas No. 3[9]	99.3	0.2–15	0.620
Pocahontas No. 4[9]	99.3	0.2–15	0.617
Illinois No. 6[10]	200	9–21	0.71

There is some evidence that the reaction of coal with oxygen takes place by addition at double-bonds and it has been pointed out that there are certain analogies in the reactions of bromine and of oxygen on coal.[13] In accord with this view is the postulation of the formation of peroxides.[14]

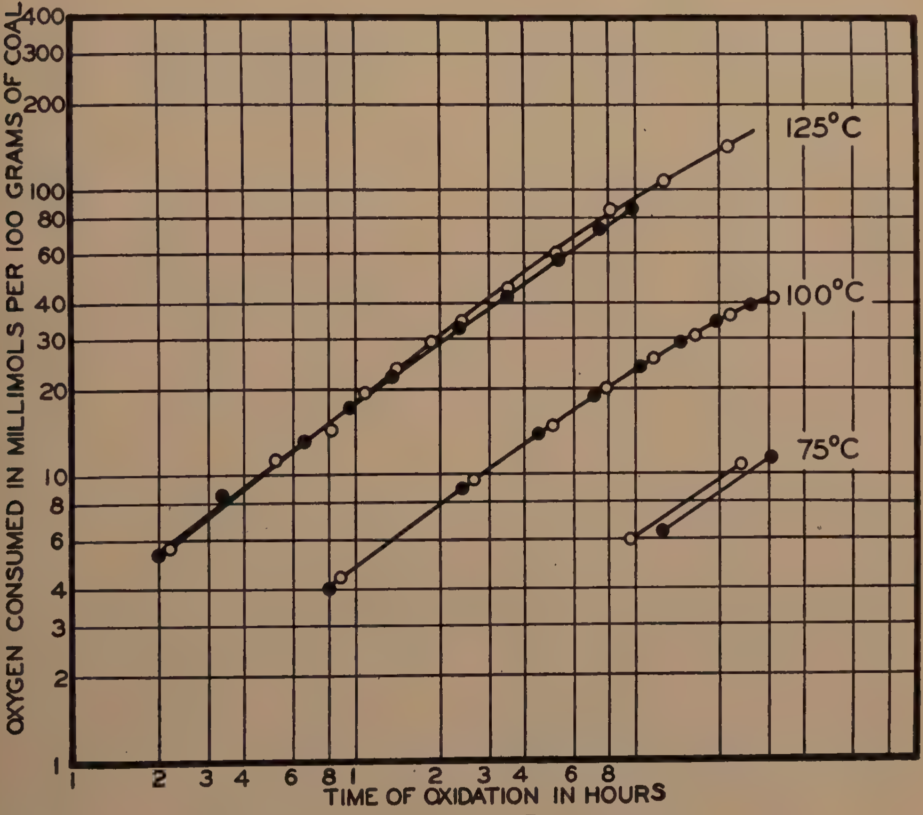

FIG 3—RATE OF OXIDATION OF POCAHONTAS COAL.

Peroxide formation, as the first step in the oxidation of coal, has been suggested by a number of workers, but specific proof of its presence has been lacking. Conventional peroxide tests have yielded negative results. It has been shown[15] that a surface layer, which oxidizes titanous chloride, is established very quickly when an Illinois bituminous coal is exposed to air. The samples of coal used in this study were collected by chipping directly from the face of the seam into a small portable ball-mill, which could be flushed by an inert gas. After pulverizing, the ground coal was transformed to closed containers in an air-tight handling box filled with an inert atmosphere. The precision of the measurements was not great, but it was sufficient to establish the general trend in the oxidizing power of the coals, that is, that unexposed coal had negligible oxidizing action and that the "active oxygen" content increased rapidly during the first few minutes of exposure to air, and more slowly thereafter.

Recently two English workers[16] have applied to coal a method developed[17] for determining peroxides in gasoline. In this method a solution prepared by dissolving ferrous sulphate and ammonium thiocyanate in a mixture of water and acetone was agitated with the coal.

Peroxides or any active form of oxygen will oxidize the ferrous salt to the ferric state, which gives the characteristic pink color of ferric thiocyanate. The pink solution was titrated with titanous chloride. Using this procedure these workers have studied the formation and disappearance of the active oxygen on several British coals. It was found that the peroxygen is relatively unstable and that it disappears rapidly at temperatures above 100°C, whereas rates of addition of oxygen and of formation of alkali soluble products, as well as cation exchange properties, increased rapidly with temperature.

TABLE 3—*Carbon-hydrogen Ratios in Coals and Oxidation Products—Reaction in Oxygen at One Atmosphere*

Coal	Time (days) and Temperature (deg C)	Atomic C/H	
		Coal	Oxidation Products
Illinois No. 6[11]	30 75°	1.22	0.10
Illinois No. 6[11]	30 100°	1.22	0.14
Durham[12]	27 107–109°	1.46	0.17
Barnsley[12]	20 107–109°	1.37	0.13
Illinois No. 6[7]	10 100°	1.27	0.17
Pittsburgh[7]	10 100°	1.29	0.15
Pocahontas[7]	10 100°	1.59	0.10

Work has been reported[7] on the low temperature reactions of oxygen with coal in a closed system in which oxygen balances could be obtained. The distribution of the oxygen in the oxidation products of three coals and at three temperatures of reaction are shown in Table 4. The outstanding feature of the data is the similarity of the distributions of the oxygen among the oxidation products for the various coals, temperatures and times of reaction. In general, somewhat over half of the total oxygen reacted, is fixed on the coal and 20 to 30 pct forms water. These oxidized coal samples, containing known amounts of fixed oxygen, were used to test various analytical methods for determining the extent of surface oxidation. Of a number of tests examined, only two, carboxyl group determination and thermal decomposition of the surface complex, showed sufficient promise to justify detailed examination. The results are illustrated in Fig 4 and 5. It is evident that only a fraction of the oxygen fixed, a few percent for the Pocahontas and 40 pct for the Illinois, reacts like a carboxyl group. A general linear relation exists, however, between the oxygen as carboxyl and that known to be present, and hence the method can be used as an analytical tool. Examination of Fig 5 shows a similar linear relation between carbon dioxide evolved from the oxidized coals, by heating in vacuum, and the oxygen fixed, and this is a more sensitive test than the former since a larger

TABLE 4—*Distribution of Oxygen*

Temperature, deg C	Time, Hrs	O_2 Used	H_2O, Per Cent	CO_2, Per Cent	CO, Per Cent	Fixed Per Cent
		Millimoles per 100 g				
Pocahontas No. 3 Coal						
75	240	10.86	30.85	11.05	1.66	56.43
75	306	11.40	29.20	11.31	1.58	57.90
100	261	39.40	31.60	12.80	2.04	53.50
100	312	40.90	32.16	12.84	1.81	53.13
125	99	86.40	29.85	13.19	2.08	54.85
125	215	143.0	27.62	14.75	2.38	55.25
Pittsburgh Coal						
75	223	36.49	18.82	14.44	1.70	65.03
75	319	48.38	21.19	9.06	1.59	68.15
100	240	104.8	27.77	12.50	2.10	57.63
100	52	45.7	19.04	10.06	1.53	69.35
125	43	139.4	17.93	15.06	2.01	65.00
125	212	316.0	27.17	18.77	2.47	51.62
Illinois No. 6 Coal						
75	246.5	49.35	18.27	13.31	2.09	66.32
75	323	68.27	20.21	12.15	1.96	65.36
100	260	189.8	33.14	18.07	2.53	46.26
100	62	106.9	24.50	13.84	2.06	59.60
125	46	275.4	34.77	16.22	3.59	45.38
125	148	327.4	29.9	19.16	2.23	48.71

fraction of the oxygen known to be present exists in a determinable form.

Many observers have noted the great difference in ease of wetting, by water, of fresh and oxidized coal surfaces. It has recently been shown[18] that the heat of wetting of an oxidized coal surface by a polar liquid like water may be twice that for a fresh surface, whereas the situation is just reversed for a non-polar liquid such as hexane. These data afford further evidence of the hydrophilic character of the primary oxidation products of coal.

Final evaluation of the commercial usefulness of the analytical methods which have been discussed must await their application to naturally aged coal samples and correlation of the analytical results with carbonization tests.

HUMIC ACID FORMATION

Relatively little information is available on the nature and rates of the low temperature reactions which are responsible for the formation of humic acids. With the higher rank bituminous coals several weeks at temperatures of 125°C may be required for conversion of 25 to 30 pct of the coal. The rate of conversion can be greatly accelerated if a liquid phase which is alkaline in reaction is present. This accelerating action presumably is caused by removal of the humic acid as a soluble salt and exposure of fresh coal surface to attack by the oxygen.

The humic acids represent the least degraded products obtained from coal, which are soluble and hence can be examined by classical methods. In the massive state they are shining black, if finely ground or in a fine suspension in water, brown. The alkaline humate solutions are probably both molecular and colloidal since a large fraction of the material readily migrates through parchment, but often significant amounts of

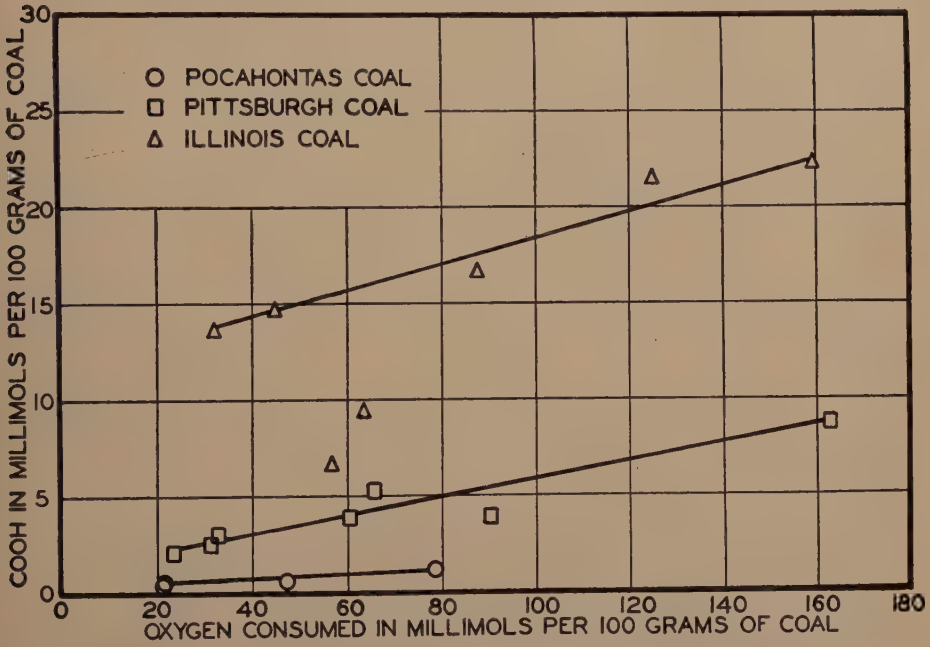

FIG 4—CARBOXYL GROUPS ON COAL AS A FUNCTION OF THE DEGREE OF OXIDATION.

nondiffusable material have also been reported. The chemical and physical evidence points to their being condensed carbocyclic and heterocyclic networks with peripheral hydrophilic acidic groups which are responsible for their alkali solubility. It seems probable that they represent fragments of the original coal network which are not significantly modified except around the edges. Further knowledge of the mechanism of their formation and their structure would be of the greatest importance in establishing the nature of coal. The problem is extremely difficult and progress is slow.

As far as is known these acids are not being prepared commercially although they are useful peptizing agents in clay slips and their ammonium salts would probably be of importance in agriculture if they could be prepared cheaply enough.

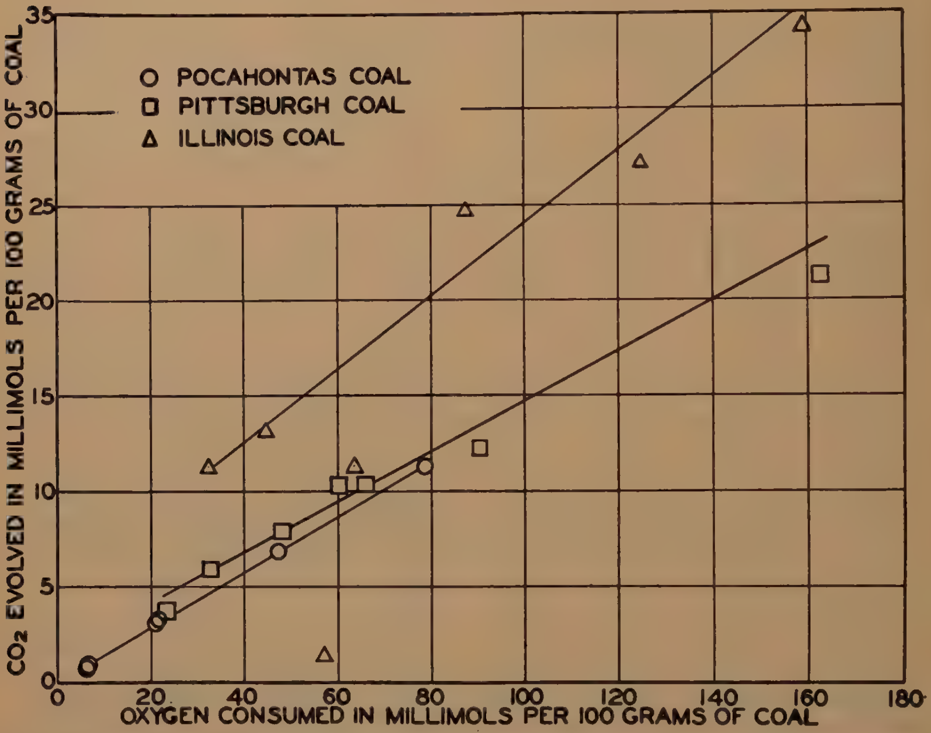

Fig 5—Carbon dioxide from decomposition of coal as a function of degree of oxidation.

Water Soluble Acids

By the action of oxygen on aqueous alkaline suspensions of finely ground coal, at elevated temperatures and pressures, the carbonaceous material of the coal can be converted completely to carbon dioxide, water and so forth, or by proper control, the reaction can be carried to any desired intermediate stage. At any given temperature, pressure and alkali concentration, the fraction of the carbon converted to carbon dioxide is a useful measure of the extent to which the desired type of product has been formed. The general course of the reaction for a Pittsburgh seam coal is shown in Fig 6. With this

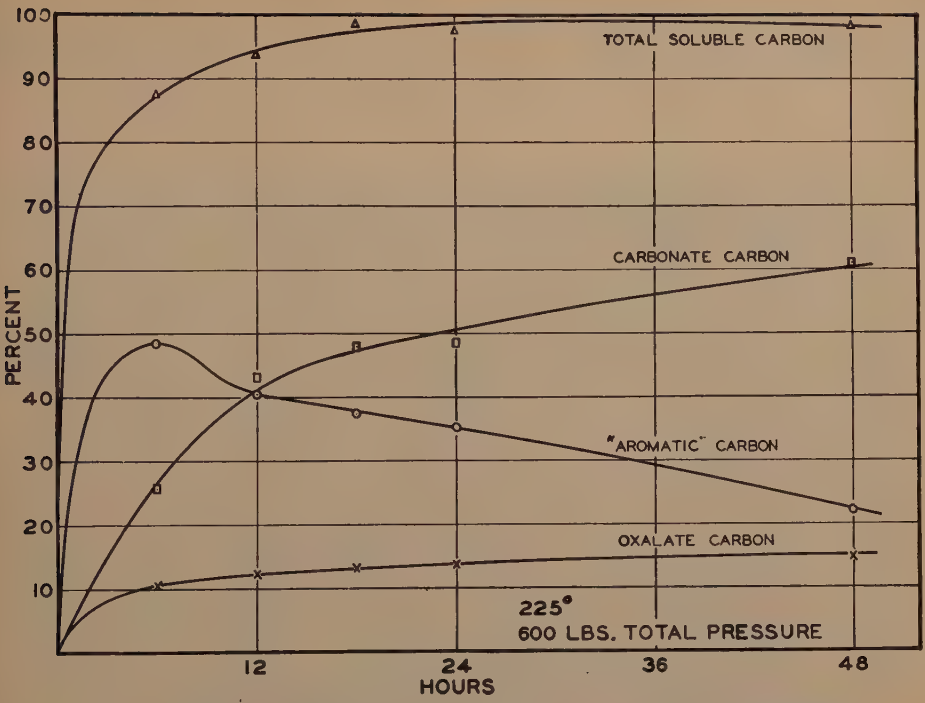

Fig 6—Rates of formation of various products in pressure oxidations.

Table 5—*Oxidation*[19] *of Aqueous Alkaline Suspensions of Carbonaceous Materials with Oxygen at Elevated Pressure*[a]

	Total Soluble Carbon, Per Cent	Carbonate Carbon, Per Cent	Oxalate Carbon, Per Cent	"Aromatic" Carbon, Per Cent	Ratio of Carbonate Carbon to Soluble Carbon	Carbonate Carbon, g	"Aromatic" Carbon, g
Illinois No. 6 seam coal	100.2	58.1	9.33	32.8	0.58	20.3	11.4
Pittsburgh seam coal	98.8	49.7	12.90	36.2	0.50	19.2	14.0
High Splint seam coal	102.2	57.7	8.03	36.5	0.56	22.7	14.3
Pocahontas No. 3 seam coal	100.6	57.0	4.50	39.1	0.52	24.2	16.6
Anthracite	100.5	59.3	3.40	37.8	0.59	24.5	15.6
500°C coke	99.5	61.3	5.20	33.0	0.62	24.5	13.2
700°C coke	85.5	59.8	4.15	21.5	0.71	24.6	8.2
High temperature coke	72.7	63.1	2.90	6.7	0.87	25.6	2.7
Graphite (40 g)	62.1	57.9	1.90	2.3	0.92	28.8	1.1
Pitch (30 g)	80.5	44.0	6.45	30.0	0.55	12.0	8.2

[a] Oxidized for 24 hr at 250°C; total pressure, 500 psi; 16 N potassium hydroxide solution; minus 200 mesh, 50-gram sample except where otherwise noted.

coal under these conditions complete transformation of the coal to the yellow, water soluble, lower molecular weight acids is only reached at the expense of converting some 50 pct of the carbon to carbon dioxide.[19] This is unfortunate, but is apparently a consequence of the condensed cyclic structure of the coal and the peripheral character of the oxidation process. It is known,[20] however, that somewhat higher yields of the water soluble acids and lower losses in carbon dioxide are obtained by the use of chemical oxidizing agents, such as potassium permanganate, at temperatures of around 100°C. It will be observed that along with carbonic acid and the water soluble, polycarboxylic aromatic acids, significant fractions of the carbon are converted to acetic and oxalic acids. The effect of rank of the carbonaceous material upon the yields of the various acids is shown in Table 5. In general, there is an increase in yield of water soluble aromatic acids up to materials of the rank of a low volatile bituminous coal. Above this point larger fractions of the carbon appear as carbonic acid, until, with graphite, the process is essentially a "combustion." In this respect the action of oxygen, at elevated temperatures, differs very markedly from that of chemical oxidizing agents which may yield as much as 38 pct of the carbon of carbon black as the polycarboxylic aromatic acid, mellitic.

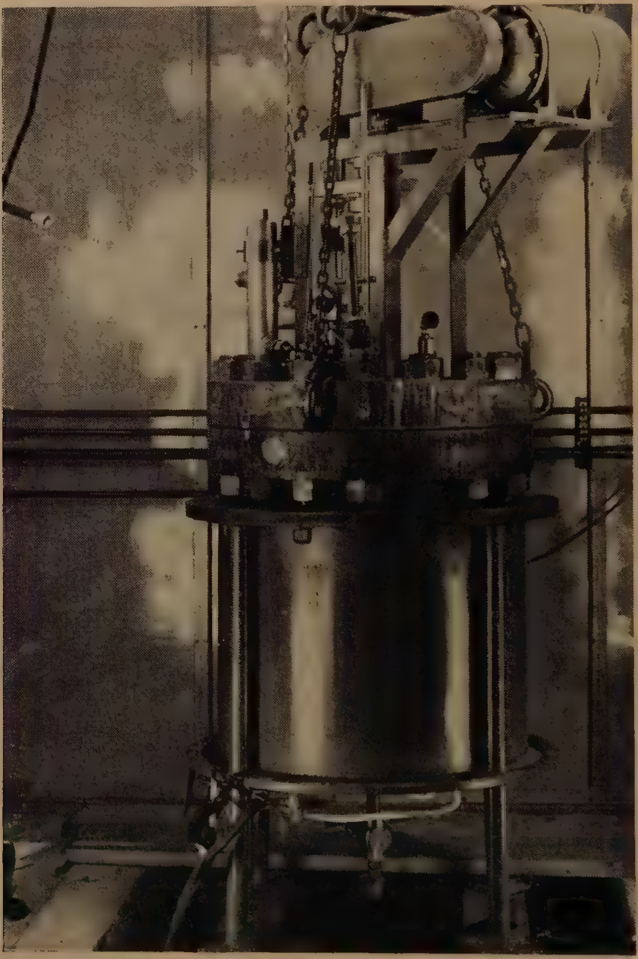

Fig 7—Reactor for pressure oxidation.

In this oxidation process the nitrogen

of the coal appears partly as ammonia and partly as acids with nuclei containing heterocyclic nitrogen. The former is readily detected in the gaseous oxidation products and nitrogen bases have been found in the decarboxylation products of the acids. The sulphur is largely converted to sulphates, but small amounts of organic sulphur are also probably present as heterocyclic nuclei in the acids.

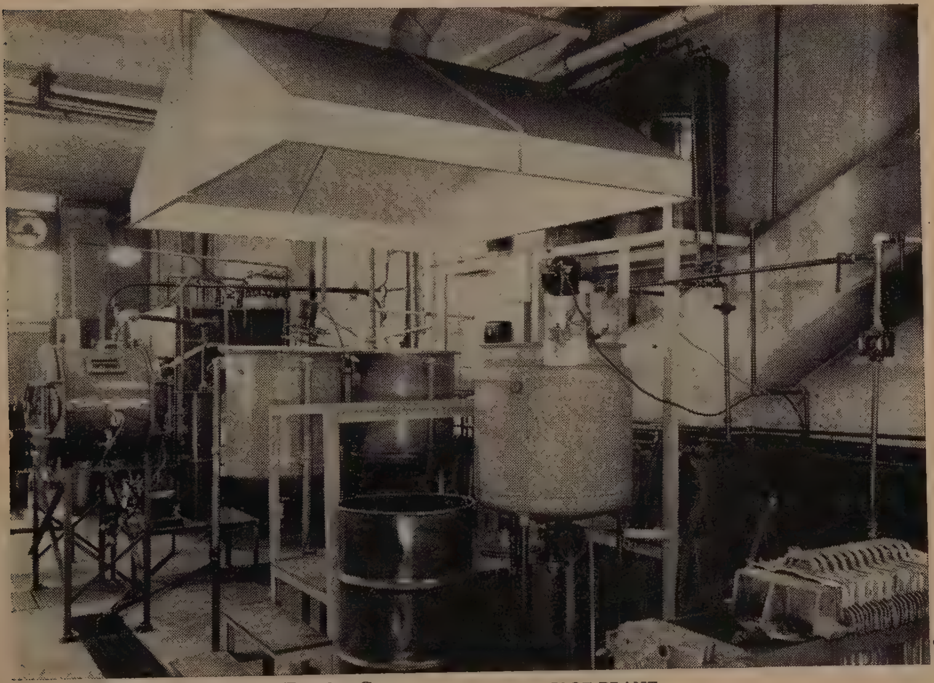

FIG 8—GENERAL VIEW OF PILOT PLANT.

The number average molecular weight of a typical sample of the water soluble, polycarboxylic aromatic acids prepared by pressure oxidation of a bituminous coal is 260, by boiling point elevation in acetone, and the average equivalent weight, determined by electrometric titration, is 80. Thus these acids have, on the average, somewhat more than three carboxyl groups per molecule. They are strong organic acids with a pH of about 2 for a 0.1 N solution. As is to be expected, a considerable range of molecular sizes is present. Fractionation of the free acids is difficult, but their esters can be fractionated in a molecular still and the separation is largely on the basis of molecular weight. The densities and refractive indices point to aromatic structures and a distribution curve for the butyl esters shows a distinct peak in the molecular weight range of 400 to 405, which corresponds to acids of molecular weight of about 220. The analytical data indicate that we do not have to deal with a series with a single type of nucleus, such as the benzene carboxylic series in which increase in molecular weight is caused entirely by addition of peripheral groups, but rather one in which increase in molecular weight is caused both by increase in nuclear size and additional number of peripheral groups per molecule.

In view of the probable industrial importance of these oxidation products of bituminous coal, a group of the sponsors of the Coal Research Laboratory, some years ago appropriated special funds for their preparation on a large scale. The pilot plant which was constructed for this purpose is shown in Fig 7 and 8. It has now been in operation for 1½ yr and numerous samples have been supplied to various industries. Although acids of this type have not hitherto been available commercially, the results so far obtained indicate that they will find a variety of uses in the formulation of plastics, as intermediates for plasticizers, as peptizing agents, as surface active compounds, and others. The enormous expansion during the past few years in the use of organic structural materials in the form of plastics, rubber, rubber-like products and fibers has emphasized the necessity for new sources of raw materials. It is believed that these products of the low temperature reactions of oxygen on coal, which can be prepared in unlimited amounts, at commercially attractive prices, will be an important factor in meeting future requirements in this field.

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Determination of Petrographic Components of Coal by Examination of Thin Sections

By B. C. Parks,* Member AIME, and H. J. O'Donnell*

(New York Meeting, February 1948)

In 1930 the late Dr. Reinhardt Thiessen set up a method of microscopic analysis and type classification of coal that has since been followed as standard practice in the coal-petrography laboratory of the Bureau of Mines. Thiessen's terminology which designated the constituent parts of coal in terms of botanical origin was introduced in 1920.[1] The technique, however, of preparing a completely representative series of thin sections from a column sample of coal for use in making quantitative determinations of coal components with a microscope was developed about 10 years later by Thiessen and coworkers G. C. Sprunk and H. J. O'Donnell.[2] Complete thin-section analyses subsequently have been made of 126 column samples of various ranks and types of American coals. The data and information were assembled from these thin-section studies and published in a large number of reports dealing chiefly with investigations of the coking and hydrogenation properties of certain American coals.

In the years following adoption of Thiessen's method, a considerable amount of work was done in collecting column samples of many coals, making routine thin-section analyses of these, interpreting the results, and finding practical application for the information in other fields of coal research. The many reports and papers covering the work in coal petrography, however, offer very little discussion that can be considered as critical appraisal of the technique. Criticism outside the Bureau most often has voiced questioning opinions concerning the accuracy of the method used of measuring coal components microscopically and classifying the coal into arbitrary types on the basis of the quantitative determinations.

The principal purpose of this paper is to discuss in some detail and present data showing the results of certain investigative work done in the petrography laboratory on the accuracy of the Bureau's method of microscopic measurement of components in coal thin sections. Data from microscopic analyses on file made by G. C. Sprunk and H. J. O'Donnell were freely used in the investigation, and a table of columnar analyses from an unpublished manuscript by Sprunk is included in this paper for reference.

In the course of the experimental work a method was developed for measuring instrumentally with a planimeter the components shown in micrographs of selected thin sections. Measuring coal components with a planimeter furnished useful information for studying the accuracy of microscopically measuring the same components in thin sections.

Petrographic Components and Type Classification of Coal

Thiessen considered coal to be composed of two visibly different major components present in various types of coal in greatly varying proportions. He named

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[1] References are at the end of the paper.

these components anthraxylon and attritus, and accepted the already well-established term fusain for designating the third and minor component.[3]

Anthraxylon

Anthraxylon was described as relatively simple in structure and essentially homogeneous in appearance. It was recognized to be in a coalified form the fairly well-preserved cellular tissues of stems, branches, twigs, roots, sporangia, and leaves which survived plant decay in the early stages of coal formation.

Anthraxylon bands are readily identified in hand specimens of coal by the conspicuously bright to brilliantly black uniformly appearing bands of this coal substance. When a layer of anthraxylon, which possesses a marked degree of brittleness, is broken the particles characteristically have thin edges, curved sides, and smooth fracture surfaces.

Anthraxylon can also be easily recognized in a thin section when seen with transmitted light under the microscope. It is present in prominent bright orange bands, sometimes shaded toward brown or red, and usually shows well-preserved cellular structure of a woody tissue seen in cross-sectional or longitudinal view.

Attritus

Attritus was defined by Thiessen as a mixture of macerated plant debris finely divided during the process of plant decay in the early peat stage and subsequently coalified.[4]

Unlike anthraxylon, the composition and structure of attritus are not simple. Moderate powers of magnification show that it is composed of many ingredients, some of which are definitely opaque when seen in the standard thin section, but most varieties are translucent and colored. The opaque ingredients of attritus—for simplicity called opaque attritus—are granular opaque matter amorphous opaque matter, and particles of fusinized tissue. Translucent attritus includes humic degradation matter, brown matter, resinous bodies, and the remains of spores, pollen, seeds, and cuticles. Some of these names suggest a little doubt as to the true botanical nature, but classification is valid because the general appearance of these ingredients in structural form and coloration remains the same in many kinds of coal.

Attrital coal is also rather easily identified in a hand specimen of coal by its dull gray color, or sometimes by a striated appearance when interbanded with fine shreds of anthraxylon. Attrital coal is a tougher substance and much more resistant to breakage than anthraxylon. When it is crushed, the resulting particles are irregularly platy in shape with the large fragments inclined to be blocky.

No particular difficulty is experienced in recognizing attritus in thin sections under the microscope. The heterogeneous mixture of ingredients of different shape, structure, translucence, and color occur in layers that are easy to distinguish from the other banded components (Fig 1).

Fusain

Fusain, the minor component of coal, is well-known and needs little description. It seems to be derived mainly from woody tissue. Fusain occurs as small particles of fusinized tissue in attritus, but is found most commonly in thin layers of more or less pure fusain interbanded with anthraxylon and attritus. The most characteristic feature of this coal component is its close resemblance to charred wood. Usually it is extremely soft and easily disintegrates into dust-size particles when roughly handled. Some fusain of the mineralized variety has considerable hardness and bright luster. All fusain is rather easily distinguished in thin sections from other opaque matter by the characteristic evidence of cellular structure.

Type Classification

Thiessen's system of classifying coal into types after the completion of a thin section analysis is based on relative proportions of components and ingredients as determined by the microscopic measurements. The five types of coal which are recognized and the critical amounts of components delimiting each type are described in Table 1.

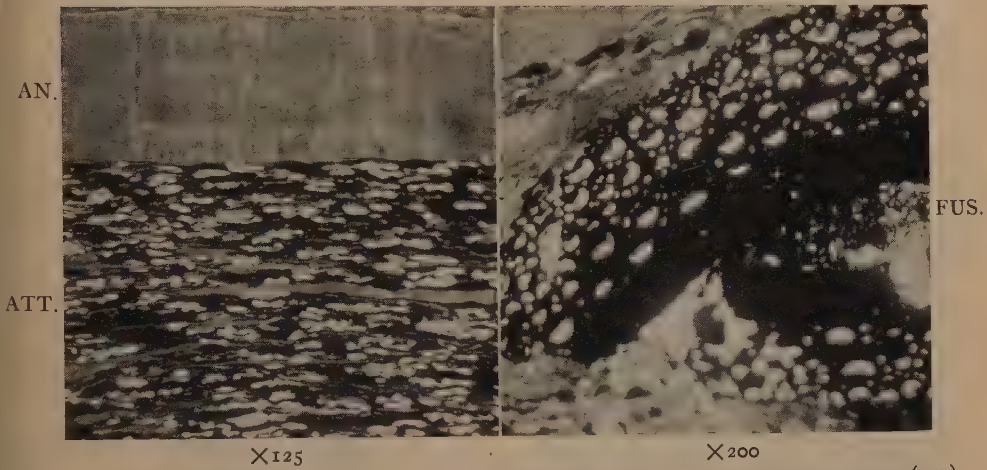


FIG 1—MICROGRAPHS OF THIN SECTIONS OF COAL SHOWING THE COMPONENTS ANTHRAXYLON (AN.), ATTRITUS (ATT.), AND FUSAIN (FUS.).

Subtype designations are used to distinguish varieties of coal that have mixed characteristics of the main types such as boghead-cannel and bright-attrital.

One of the objects of this study was to learn whether the microscopic measurement of coal ingredients was sufficiently exact to justify establishing type classifications within arbitrary limits such as set up in Table 1.

MICROSCOPIC ANALYSIS OF COAL IN THIN SECTIONS

Preparing Thin Sections

The first step in preparation for a complete coal thin-section analysis is the task of getting a column sample of the coal to be studied. It is quite desirable to have a uniform pillarlike sample representing the entire bed from roof to the floor and approximately 12 in. square in cross-section. The hard, brittle nature of most coals makes column sampling a very difficult job, and near perfect columns are secured only with considerable care from the softer coals of lignite rank (Fig 2). When the petrographic analysis is to be made in conjunction with other tests and studies of the same coal, the column sample is cut from the coal bed in adjacent position to the samples removed for the other tests.

TABLE 1—*Type Classification*

TYPES	CRITICAL AMOUNTS OF COMPONENTS
Cannel coal........	Less than 5 pct anthraxylon and predominantly translucent attritus with little or no oil algae
Boghead coal......	Less than 5 pct anthraxylon and the translucent attritus predominantly oil algae
Bright coal........	More than 5 pct anthraxylon and less than 20 pct opaque attritus
Semisplint coal.....	More than 5 pct anthraxylon and 20 to 30 pct opaque attritus
Splint coal.........	More than 5 pct anthraxylon and more than 30 pct opaque attritus

The column sample, after being brought to the laboratory, is put under a cut-off saw and a subcolumn approximately 3 in. wide and 6 in. deep cut out. With the bedding planes of the coal in mind, the subcolumn is cut so that the narrow, vertical surface can be oriented in a parallel position to the sampled face of the coal bed. Beginning at the bottom and continuing to the top, parallel lines spaced 0.8 in. apart are marked off on the subcolumn. These lines mark the position of the small blocks which are cut from the subcolumn and used in making the thin sections (Fig 3). If a profile analysis is to be made, a duplicate subcolumn is also cut, and the surface to be studied in detail is polished to a mirror-like finish. Care is exercised in cutting both subcolumns to have the working faces of each adjacent to the other. This is done to facilitate a comparative study, if so desired, of the polished surface profile analysis (megascopic) and the thin-section analysis (microscopic).

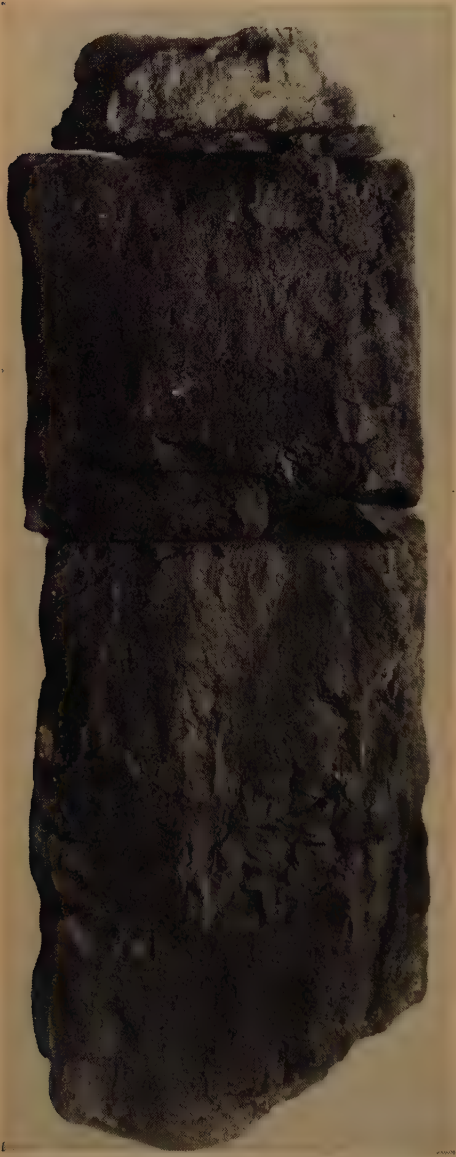


Fig 2—Photograph of a column sample. Lignite from Ouachita Co., Ark. Height 39 in., width and depth 14 in., weight 230 lb.

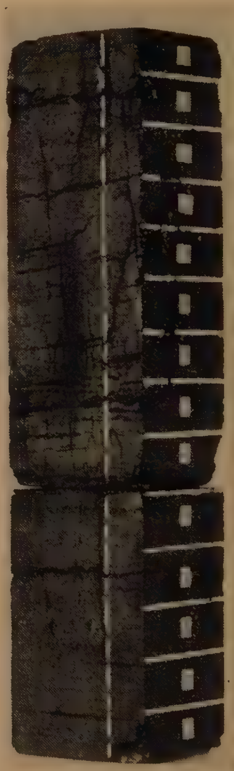


Fig 3—Showing how small blocks to be used in preparing thin sections are marked off for sawing in serial position on subcolumn.

Small blocks cut from the subcolumn as

indicated by the spaced lines in Fig 3 are reduced in size by cutting to dimensions about 1 in. long, 0.8 in. wide, and ½ in. thick. One face is then smoothed and polished and after first being properly oriented is cemented to a standard glass slide. The mounted blocks are then ground down to a thickness of about 5 to 10 microns. This is done with exceeding care to produce a thin slice of coal as nearly as possible of uniform thickness, which will transmit light through all the coal components except the truly opaque. The entire procedure of making thin sections requires great skill during each operation. The technique cannot be adequately described in a few words nor expert skill in its performance acquired in a short time.[5]

The completed thin section ready for use consists of a standard microscope slide to which the thin slice of coal approximately 0.8 in. wide and 1 in. long is mounted securely with a cement having the same optical properties as the glass. The section is protected with a standard glass cover slip cemented to the section with Canada Balsam. The complete series of thin sections prepared from the column can be thought of as a disconnected, transparent, ribbonlike sample of coal about 8 microns thick and 1 in. wide, representing an entire coal bed or some sizable part.

The ribbonlike sample of thin sections is not, however, a complete representation of the coal bed because of some loss from sawing and polishing. Coal lost in preparing sections is kept at a surprisingly low figure for most coals by careful work. It is not unusual to have the measured widths of all sections total 90 to 95 pct of the height of the original column sample. The amount of coal lost in thin section preparation because of difficulty in grinding to right thinness increases as the coal becomes higher in rank up to the low-volatile bituminous coals. It is not practical to make thin sections of anthracite since this coal does not become translucent at any thinness to which the section can be successfully ground.

Microscopic Measurement

The basic instrument used in measuring coal components in a thin section is a biological-type microscope equipped with binocular eyepieces. A pair of 15-power oculars is used with 32- and 16-mm objective lenses to obtain working magnifications of 60 and 150 diam.

Into one of the eyepieces is inserted a Whipple micrometer disc. This is a glass disc on which is centered a 7-mm square field divided into one hundred 0.7-mm squares with the central one subdivided into twenty-five 0.14-mm squares. These values represent the real dimensions of the square fields scribed on the Whipple disc (Fig 4). Under microscopic magnification the dimensions change; therefore, it is necessary to calibrate the disc with a stage micrometer for different powers of magnification as is shown in Table 2. In standard practice opaque attritus, fusain, and mineral matter are measured at a magnification of 60 diam and anthraxylon at 150 diam. Translucent attritus is not measured directly but is determined by difference.

TABLE 2—*Calibrated Values of the Whipple Disc with Binocular Microscope**

Magnification	Central Vertical Field (10 Squares)	One Square (⅒ C.v.f.)	One Subsquare (⅕ Square)
X	Mm	Mm	Mm
0	7	0.7	0.14
40	2.78	0.278	0.056
60	1.86	0.186	0.037
100	1.09	0.109	0.022
150	0.73	0.073	0.015
200	0.56	0.056	0.011
300	0.36	0.036	0.007
475	0.23	0.023	0.005
712	0.15	0.015	0.003

* Calibrated values of the Whipple disc made with binocular microscope by different observers vary slightly when horizontal adjustment (interpupillary spread) of the eyepieces is necessary.

The technique of measuring coal in the thin section is accomplished by manipulating the mechanical stage of the microscope to move the central vertical field of the Whipple micrometer disc along two lines extending from bottom to top of the section and located near the middle of the right and left half of the section. Only the central vertical tier of squares containing the subdivided midsquare of the disc is used in measuring the coal components that are crossed in transecting the section. For visual convenience the remainder of the disc is masked off. In manipulating the mechanical stage during measurement the vertical movement along the transect is not continuous but is done in steps. Each step moves the thin section a distance which is the length of the central field of the Whipple disc. When the total areas of a coal component or components lying under the unmasked central field are determined, the section is moved along the transecting line exactly the distance of the central field. These field moves are thus continued until the section is crossed. From 20 to 25 field moves are necessary to complete a single transect across the section varying of course with width of section and magnification. A shift of the stage horizontally brings the section into position for measuring the next transect (Fig 4). Values determined from the two transect determinations are recorded separately and the averages of the two are the accepted figures for the section.

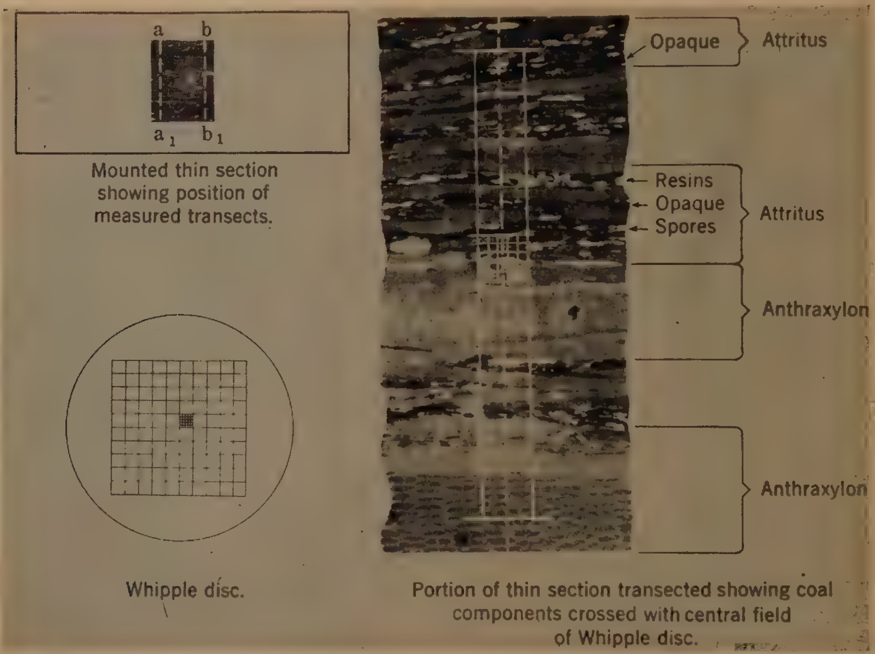

a
b
a₁
b₁
Mounted thin section showing position of measured transects.
Whipple disc.
Opaque
Attritus
Resins
Opaque
Spores
Attritus
Anthraxylon
Anthraxylon
Portion of thin section transected showing coal components crossed with central field of Whipple disc.

FIG 4—DIAGRAMMATIC SKETCHES ILLUSTRATING MICROSCOPIC MEASUREMENT OF COAL COMPONENTS IN A THIN SECTION.

After all the thin sections representing the entire column sample have been measured as described, the resulting data are assembled and set up graphically in columnar form as shown in Fig 5. Percentagewise distribution of anthraxylon, translucent attritus, opaque attritus, and fusain throughout the column is shown by bar diagrams in which each bar represents a thin section. On the basis of relative percentages of components the coal is classified into types. The coal types usually are formed into distinct layers which are

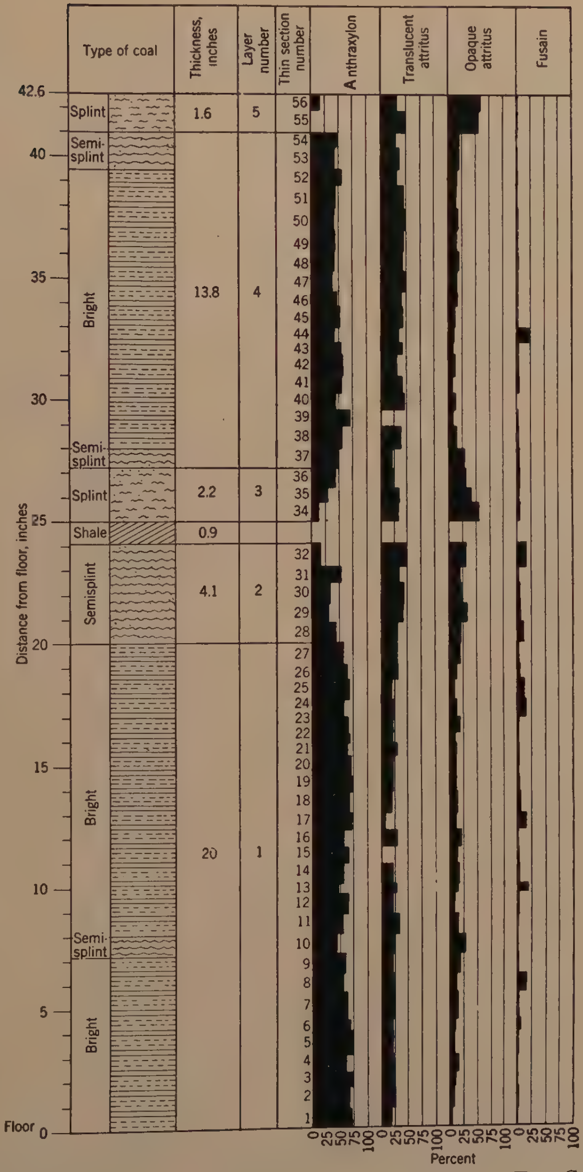


Fig 5—Components and types of coal in Powellton Bed Coal. Elk Creek No. 1 Mine, Logan Co., W. Va.

easily discernible when the coal column is examined megascopically. Distribution of components in the various type layers and the average composition of the entire column, (Fig 5), is shown in Table 3.

TABLE 3—*Components in Type Layers, Per Cent*

Layer	Type	Sections	Anthraxylon	Translucent Attritus	Opaque Attritus	Fusain
1	Bright	1–27	62	23	11	4
2	Semi-splint	28–32	34	38	22	6
3	Semi-splint	34–36	28	29	40	3
4	Bright	37–54	48	37	12	3
5	Splint	55–56	5	40	54	1

Average Composition of Entire Column, Per Cent

Anthraxylon	Translucent Attritus	Opaque Attritus	Fusain
51	30	15	4

ACCURACY TESTS OF STANDARD METHOD OF THIN-SECTION ANALYSIS OF COAL

The question of whether or not significant variations occur in the results of microscopic analysis by different observers or with the same observer using different powers of magnification has been the subject of much discussion and some experimentation in the coal petrography laboratory. One such test had seemed to show that with the use of increasing magnifications the amount of opaque matter as determined in the same thin section progressively decreased.

Effect of Increased Magnification

A simple experiment was first carried out to test whether increasing the magnification had any effect in determining the results of measuring the same opaque areas. A number of parallel lines of variable width were ruled with black India ink on a standard glass slide and a thin vertical white line of fixed length was drawn through these to serve as a transect guide. The total amount of opaque area was determined in crossing the inked lines along the transect line. Three operators made independent determinations with two microscopes of different make and using magnifications of 40, 100, 200, 475, and 712 diam.

The data from this experiment shown in Table 4 established the fact that, provided the contrast between opaque and translucent areas was well defined, a high degree of accuracy could be obtained by measuring at any magnification and with different make microscopes when careful work was done. A tendency toward a slight cumulative error was noted as the number of moves of the disc micrometer increased

TABLE 4—*Comparison of Microscopic Measurements Made with Whipple Disc of Ruled Opaque Lines*

Mag.	Unit of Meas.*	Operator B			Operator C			Operator D		
		No. Field Moves	Length of Trans.	Per Cent Opaque	No. Field Moves	Length of Trans.	Per Cent Opaque	No. Field Moves	Length of Trans.	Per Cent Opaque
X	Mm		Mm			Mm			Mm	
40	2.78	7.0	19.5	27.0	7.1	19.7	27.7	7.1	19.7	27.6
100	1.09	17.7	19.3	27.8	17.8	19.4	28.1	17.9	19.5	28.3
200	0.56	34.6	19.4	28.0	35.0	19.6	28.4	34.8	19.5	28.1
475	0.23	83.4	19.2	28.2	83.6	19.2	28.4	83.5	19.2	28.0
712	0.15	126.2	18.9	26.3	126.5	19.0	27.4	126.8	19.0	27.6
Averages...				27.5			28.0			27.9

* Calibrated length of central vertical field of the Whipple disc.

with higher magnifications. This was noticed in apparently a slight shortening of the measured length of the transect line with increasing number of necessary field moves, but no tendency toward cumulative error was noted in the results of the total opaque determinations.

Planimeter Measurement of Opaque Matter

The measurement of coal components by use of the microscope and micrometer disc is essentially a highly refined method of visual estimation. To determine how good or how bad the results of microscopic estimations by a trained observer really are, it would be desirable to check the microscopic determinations against values obtained by some different method of measuring. The use of a polar planimeter and enlargements of thin-section micrographs for measuring coal components to compare with microscopic determinations seemed to offer interesting possibilities.

An experiment embodying this idea was carried on in the following manner: Micrographs were made of two coal thin sections which contained relatively high percentages of opaque matter. In one section the opaque matter occurred in more or less random distribution, in the other it was distinctly banded. The micrographs were enlarged 3¾ times raising the magnification to approximately 750 diam. Each micrograph was trimmed to a size 227 mm high by 180 mm wide and divided with vertical lines into 18 strips equal in width (Fig 6).

For the purpose of the experiment the width of the trimmed micrograph (180 mm) was assumed to be the width of the central field of the micrometer disc at 40 magnifications. By computing the ratio of the two magnifications the width of the central field at × 712 was found to be 1⁄18 as wide as at × 40. Thus in changing magnification from 40 to 712 power the width of the area measured with the central field is reduced to a width 1⁄18 that of the low magnification. Therefore by design the test involved measurement of areas marked off on a micrograph that had the same comparable ratio of size as the microscopic fields measured in the thin section at × 40 and × 712 magnifications.

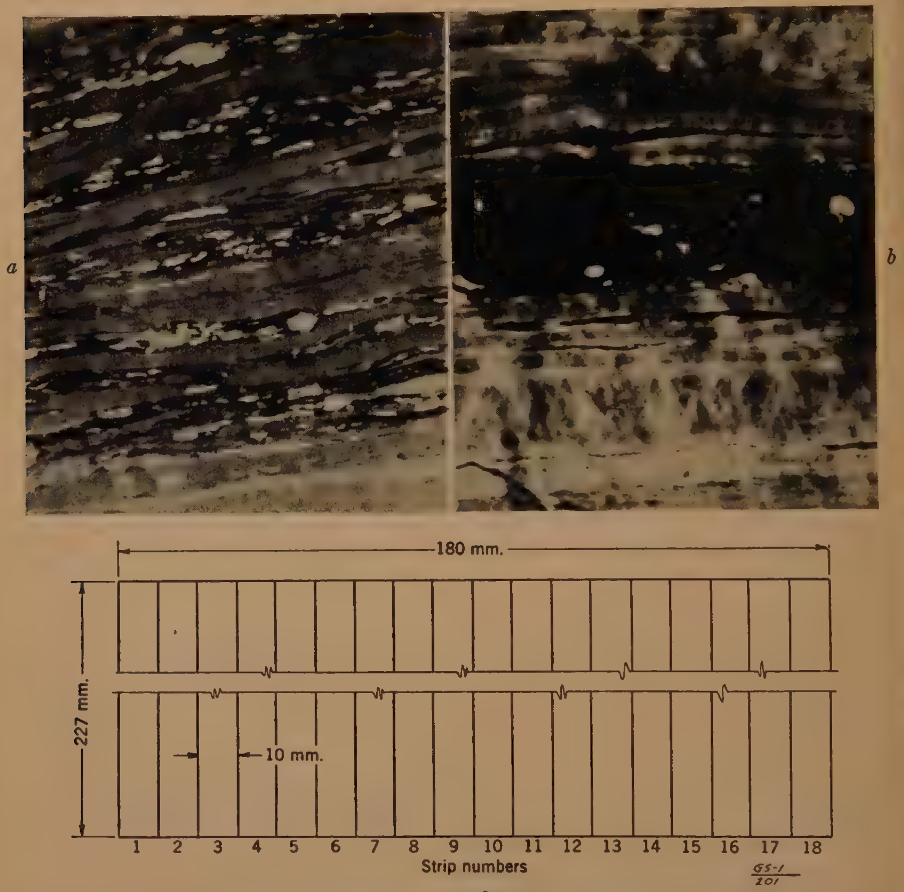
Before actual measurements of the opaque areas on the micrographs with the planimeter was begun, considerable practice work was done measuring various shaped small areas drawn on glossy cardboard. This was done to give proficiency in manipulating the instrument and to determine its limitations in regard to accuracy. It was found that with careful work the planimeter was quite reliable for measuring areas of only a few square millimeters which were comparable in size and outline to the small opaque areas of the enlarged micrographs.

All black areas on the micrograph enlargements were checked with the original thin section under the microscope to make certain that only opaque attritus was represented. The confirmed opaque areas were then outlined with a medium-hard pencil to form a track that could be easily followed with the tracing pointer of the planimeter. This feature gave assurance that the same area would be traced each time in making repeat measurements for checking purposes even when done by different operators.

In securing the experimental data, every strip on both photographs was measured and the results recorded separately. The data thus obtained were then subjected to a certain amount of analytical study. Tables 5 and 6 summarize the information and show some of the more important results.

The averaged value of the 18 measured strips of the Millers Creek coal (Fig 6A) is shown to be 14.2 pct. This value was accepted as representing the total opaque area measured in the photograph. An approximately identical value was obtained by dividing the sum of all the individually measured opaque areas in the 18 strips by the total area of the micrograph. The value

14.2 is in surprisingly close agreement with the previously made microscopic determination of the same area in thin section which was 14.0 pct.



Strip pattern drawn on enlargments (X3$\frac{3}{4}$) of photomicrographs (A) and (B).

FIG 6—SHOWING AREAS OF TWO THIN SECTIONS PHOTOGRAPHED AND ENLARGED FOR PLANIMETER MEASUREMENT OF THE OPAQUE AREAS.

A close examination of the values in column 1 of Table 5 shows a range from a low of 6.4 to a high of 20.3 pct. This range can be expressed also as a variation from the average value within extremes of +6.1 and −7.8 percentage points. The range of deviation therefore equals 13.9 percentage points. The deviation of each strip from the average was computed as shown in column 2; these values were squared (column 3), and with these data the standard deviation (3.7) was computed by the formula $\sigma = \sqrt{\frac{(\Sigma\chi^2)}{N}}$.

Data from the measurement of opaque areas in micrograph B (Fig 6), representing banded opaque attritus in the Pittsburgh coal are shown in Table 6. The results of this test show the same general trend, but with slightly less deviation from the average in amount of opaque measured in each strip. The value of 31.1 pct opaque computed by averaging the 18 measured strips was almost the same amount obtained by the microscopic determination, 30.8 pct.

The experimental tests of measuring opaque areas outlined in micrographs of thin sections gave results that were in extremely close agreement with microscopic determinations of opaque attritus in the areas of the thin sections that were photographed. This close check between visual microscopic estimation with an eyepiece micrometer and measurements made with a manually operated instrument seemed to lend positive support for reliability of microscopic determinations.

TABLE 5—*Results of Planimeter Measurement of Opaque Matter in Micrograph of Thin Section of Millers Creek Coal*

Strip No.	1 Per Cent Opaque	2 Deviation from Avg. (X)	3 X^2
1	20.3	+6.1	37.21
2	12.7	−1.5	2.25
3	13.5	−0.7	0.49
4	17.4	+3.2	10.24
5	12.0	−2.2	4.84
6	12.9	−1.3	1.69
7	11.1	−3.1	9.61
8	11.1	−3.1	9.61
9	14.0	−0.2	0.04
10	19.7	+5.5	30.25
11	17.4	+3.2	10.24
12	12.7	−1.5	2.25
13	17.1	+2.9	8.41
14	18.7	+4.5	20.25
15	16.3	+2.1	4.41
16	13.7	−0.5	0.25
17	8.8	−5.4	29.16
18	6.4	−7.8	60.84
	255.8	54.8	242.04
Average....	14.2		

Extremes of deviation about average value... (+6.1) (−7.8)
Range of deviation between extremes........ 13.9
Mean deviation.......................... 3.0
Standard deviation...................... 3.7
Microscopic determination same area in thin section.............................. 14.0

The study also showed that if the total area under observation was divided into a number of equal strips, each of which was measured, these strips could be considered incremental parts of the whole area, and certain quantitative relationships between the increments and the whole could be determined by analytical study.

Microscopic Measurement of Opaque Matter in Block Sample

With the information obtained from the previously described work as background, a statistical study of the measurement of opaque matter was extended to the thin sections of a block sample of Pittsburgh coal. The block sample selected had been thin sectioned and microscopically analyzed in 1941. The suite of 10 thin sections made from the sample was still available in good condition for study as also were the records of the analysis.

TABLE 6—*Results of Planimeter Measurement of Opaque Matter in Micrograph of Thin Section of Pittsburgh Coal*

Strip No.	1 Per Cent Opaque	2 Deviation from Avg. (X)	3 X^2
1	31.7	+0.6	0.36
2	28.5	−2.6	6.76
3	28.4	−2.7	7.29
4	29.3	−1.8	3.24
5	27.3	−3.8	14.44
6	26.6	−4.5	20.25
7	29.5	−1.6	2.56
8	31.0	−0.1	0.01
9	28.6	−2.5	6.25
10	28.7	−2.4	5.76
11	27.7	−3.4	11.56
12	31.4	+0.3	0.09
13	32.8	+1.7	2.89
14	37.4	+6.3	39.69
15	35.7	+4.6	21.16
16	34.7	+3.6	12.96
17	34.2	+3.1	9.61
18	36.0	+4.9	24.01
	559.5	50.5	188.89
Average....	31.1		

Extremes of deviation about average value... (+6.3) (−4.5)
Range of deviation between extremes........ 10.8
Mean deviation.......................... 2.8
Standard deviation...................... 3.3
Microscopic determination same area in thin section.............................. 30.8

It was planned in this test that the microscopic measurements would include a complete coverage of all 10 sections, accomplished as follows: Beginning at the left margin of each section a vertical strip the width of the central field of the micrometer disc at 40 magnifications was measured, and in successive order strips the width of this field were measured to the right edge of the section. The number of strips measured in each thin section ranged from 40 to 82, depending on the length of the section. Fusain was included in the measurement of the opaque attritus.

The significant results obtained in this test are summarized in Table 7. The values shown in the table opposite *A* represent the earlier microscopic determination by the standard method of opaque attritus and fusain in this particular block sample of Pittsburgh coal. The records of the analysis do not state the exact position in each section the transects were taken, but no doubt the practice was followed of transecting near the center of the right and left half of the section. The values beneath and opposite *B* in Table 7 represent complete strip coverage of each section and can be assumed to be the closest obtainable approximation to absolute values by this method of microscopic measurement.

Data representing strip coverage of each section from approximately 2 to 20 pct of the total area, obtained by equally spacing the transects at various interval widths across the sections, are included in the table. This arrangement of the data is, in effect, a sampling analysis of the strip measurements of the 10 sections.

If the values in Table 7 opposite *B* (line marked *percent opaque*) are accepted as the correct amount of opaque attritus and fusain in each thin section, then similar values in the rest of the table show the variation from the true amount as consequence of incomplete coverage of the thin sections. It will be noted in the table that the values for *B*-1, representing 20 pct coverage by the space sampling as described, differ very little from those for 100 pct coverage. The variation for each section from the true value for the section represented by complete coverage became more erratic as the percentage of coverage decreased.

The quantity of opaque attritus and fusain in the block sample determined by averaging the measurements of the 10 thin sections was essentially the same irrespective of the degree of coverage. This is shown in the last vertical column under the

TABLE 7—*Tabulated Data of Microscopic Determination of Opaque Attritus and Fusain in 10 Thin Sections Representing Block Sample of Pittsburgh Coal*

		1	2	3	4	5	6	7	8	9	10	Total	Averages
A	No. of transects	2	2	2	2	2	2	2	2	2	2	20	
	Per cent coverage	2.7	3.2	3.2	5.0	2.8	2.4	3.1	2.5	3.4	4.8		3.3
	Per cent opaque	19.0	10.0	11.0	9.0	9.0	11.0	16.0	13.0	18.0	18.0		13.4
B	No. of transects	73	63	62	40	72	82	65	79	59	42	637	
	Per cent coverage	100	100	100	100	100	100	100	100	100	100		100.0
	Per cent opaque	17.6	7.4	11.0	10.6	8.1	11.8	17.2	14.0	16.7	22.1		13.7
B-1	No. of transects	15	13	13	8	14	17	13	16	12	8	129	
	Per cent coverage	20.5	20.6	20.9	20.0	19.4	20.7	20.0	20.3	20.3	19.0		20.2
	Per cent opaque	17.1	7.5	11.0	10.9	8.0	11.6	17.2	13.9	16.5	22.9		13.7
B-2	No. of transects	7	7	7	4	8	9	7	8	6	5	68	
	Per cent coverage	9.6	11.1	11.3	10.0	11.1	11.0	10.8	10.1	10.2	11.9		10.7
	Per cent opaque	17.7	7.4	11.5	10.4	7.5	11.6	17.0	15.6	16.8	21.7		13.7
B-3	No. of transects	4	4	4	2	4	5	4	4	3	2	36	
	Per cent coverage	5.5	6.3	6.5	5.0	5.6	6.1	6.2	5.1	5.1	4.8		5.6
	Per cent opaque	18.6	7.6	12.2	10.4	8.3	11.3	17.1	14.6	16.4	21.6		13.8
B-4	No. of transects	2	2	2	2	2	2	2	2	2	2	20	
	Per cent coverage	2.7	3.2	3.2	5.0	2.8	2.4	3.1	2.5	3.4	4.8		3.3
	Per cent opaque	16.4	6.6	9.7	10.4	9.0	12.6	17.2	13.8	16.3	21.6		13.4
B-5	No. of transects	1	1	1	1	1	1	1	1	1	1	10	
	Per cent coverage	1.4	1.6	1.6	2.5	1.4	1.2	1.5	1.3	1.7	2.4		1.7
	Per cent opaque	19.8	8.0	12.9	11.0	7.7	12.4	16.0	14.1	15.7	18.2		13.6

A Previous analysis by standard method.
B Complete coverage of every thin section.
B-1 Beginning with 3d, every 5th transect.
B-2 Beginning with 5th, every 10th transect.
B-3 Beginning with 10th, every 20th transect.
B-4 Center transect in right and left half of section.
B-5 One transect sampled through center of section.

heading averages. The range of deviation for the seven values given in this column is 0.4 percentage point and the maximum deviation from the 100 pct coverage is 0.3 percentage point.

The results given in Table 7 seem to show that the determinations of total quantity of opaque components were essentially the same, whatever the amount of transect coverage for a section within a range of 1.7 to 100 pct. The term "coverage" as used refers to the number of transects made in relation to the total possible number for the particular section and the magnification used. The percentage values of coverage can also be expressed in terms of areal or linear measurement, inasmuch as thin sections are very closely rectangular.

It is probably best to think of microscopically transecting thin sections as a form of linear volumetric measurement in which the total linear distance transected is divided into equal units the width of the section and spaced equally across the section. Considered as increment sampling, it is suggested by the data that if the number of increments sampled represents about 20 pct of the possible number for a section the resulting determination will approximate that obtained by complete section coverage with a variance not exceeding 0.8 of a percentage point. Deviation from the values obtained with 100 pct coverage for the 10 thin sections determined by increment transect sampling representing from 20.3 to 1.7 pct coverage of the sections is shown in Table 8.

TABLE 8—*Deviation of 10 Sections from Complete to Partial Coverage*

Thin Section Coverage, Per Cent	Average Deviation	Extremes of Deviation		Range of Deviation
100.0	0	0	0	0
20.2 (avg. 10 sect.)	0.02	−0.5	+0.8	1.3
10.7 (avg. 10 sect.)	0.04	−0.6	+1.6	2.2
5.6 (avg. 10 sect.)	0.05	−0.5	+1.2	1.7
3.3 (avg. 10 sect.)	0.06	−1.3	+0.9	2.2
1.7 (avg. 10 sect.)	0.12	−3.9	+2.2	6.1

The second column above shows that the average deviations from complete to partial coverage of 10 sections are insignificantly small. This suggests the reason why the minimum number of transects sampled in each section was adequate to evaluate the opaque components in the total block sample of 10 sections within an error of 2 pct. However, to achieve approximately the same degree of accuracy for any particular thin section of the series, it was necessary to sample about 20 pct of the possible number of transects in the section.

The block sample of coal under study was classified on the basis of complete analysis of all components as a finely banded bright coal with the components fairly uniformly distributed. A minimum number of increment transects would seemingly suffice to evaluate the components in coal of this general petrographic composition.

Measurement of Anthraxylon in Block Sample

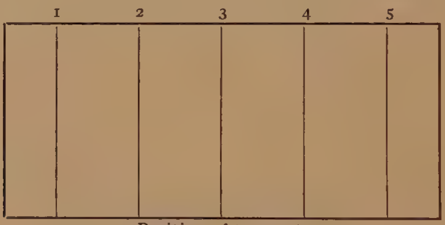
Using the thin sections from the same block sample of Pittsburgh coal, similar tests were applied to measuring the anthraxylon content. These determinations differed from the previous experiment with opaque matter in that a limited number of spaced transects were sampled instead of complete coverage of each thin section. The summarized results of the test are shown in Table 9. A sketch showing the position of the transecting lines in each section is included at the bottom of the table. The same relative spacing, determined by measured lateral moves, was applied to each of the 10 sections.

It is probably reasonable to assume that the values under column 8 of Table 9, which are averages of 5 transects in each section, represent the closest approximations in the data given to the total quantity of anthraxylon in these sections. Also, the grand average at the bottom of this same column can be accepted as the determined amount of anthraxylon in the block sample.

A study of the values in column 8 compared with the rest of the data in the table makes it evident that conclusions to be drawn from these experimental data follow the same pattern as for the similar experiment in measuring opaque matter.

Table 9—*Tabulated Data of Microscopic Determination of Anthraxylon in* 10 *Thin Sections Representing Block Sample of Pittsburgh Coal*

Section No.	1	2	3	4	5	6	7	8	9
	Tran. 1	Tran. 2	Tran. 3	Tran. 4	Tran. 5	Avg. of Trans. 1 and 5	Avg. of Trans. 2 and 4	Avg. of 5 Trans.	Standard Method
1	25.7	23.2	25.0	27.7	20.4	23.1	25.5	24.4	26.0
2	51.9	48.3	47.7	49.6	44.6	48.3	49.0	48.4	50.0
3	41.3	44.6	42.3	47.3	44.6	43.0	46.0	44.0	46.0
4	60.0	60.0	64.6	61.0	63.6	61.8	60.5	61.8	61.0
5	56.0	56.2	59.0	56.8	58.1	57.1	56.5	57.2	53.0
6	45.7	46.1	41.4	49.8	44.3	45.0	48.0	45.5	45.0
7	49.5	50.0	53.7	53.3	51.5	50.3	51.7	51.5	51.0
8	58.2	60.0	57.8	53.2	52.7	55.1	56.6	56.4	55.0
9	46.2	43.7	45.2	43.6	52.5	49.4	43.7	46.2	49.0
10	43.5	46.9	44.2	41.7	44.3	43.9	44.3	44.1	45.0
Averages....	47.8	47.9	48.1	48.4	47.6	47.7	48.2	48.0	48.1



Position of transects

Measurement of Opaque Attritus and Fusain in Column Sample

The final experimental test dealing with microscopic measurement of coal components involved determination of opaque attritus and fusain by complete coverage of each section in a series representing an entire column sample. A relatively short column from a 3½-ft coal bed and represented by 34 prepared thin sections was selected. The coal had been classified on the basis of an earlier analysis as semisplint. The high content of opaque attritus and fusain made the test rather rigorous, particularly in comparing results with the former analysis. In thin sections high in opaque matter the quantity of dark-brown matter which is probably of transitional nature is increased. With this increase the factor of personal judgment in classifying some of this transitional material either as brown matter or opaque attritus may have a marked effect on the quantitative results.

If the section values in Table 10 under columns *A* and *B* are compared, it will be noted that the greatest deviation from duplication of the two usually occurs in thin sections with high percentages of opaque attritus. These are the types of thin sections in which correct thinness to be certain of translucence is most difficult to obtain. Although the determinations represented by columns *A* and *B* were done by two different operators almost 10 years apart, it can be seen that the evaluation of opaque attritus and fusain in the column as given by the averages at the bottom of the columns are in good agreement. If the average for the complete coverage determination column *B* is accepted as the correct value then the average of two transects measured in each section shown in column *A* gives a value only 7.5 pct in variance.

TABLE 10—*Comparison of Microscopic Determinations of Opaque Attritus and Fusain in Column Sample of Eagle Coal from West Virginia by Sampling Two Transects in Each Section (Standard) and Complete Transect Coverage*

Thin Section	A (Standard)	B (Complete Coverage)	Differences A − B
1	18.0	13.1	− 4.9
2	4.0	3.1	− 0.9
3	29.0	24.5	− 4.5
4	28.0	28.4	+ 0.4
5	8.0	8.7	+ 0.7
6	29.0	24.8	− 4.2
7	14.0	10.3	− 3.7
8	19.0	14.0	− 5.0
9	22.0	17.1	− 4.9
10	30.0	28.5	− 1.5
11	29.0	27.5	− 1.5
12	62.0	46.9	−15.1
13	45.0	45.2	+ 0.2
14	22.0	25.7	+ 3.7
15	46.0	43.1	− 2.9
16	19.0	17.5	− 1.5
17	14.0	14.2	+ 0.2
18	36.0	31.4	− 4.6
19	36.0	35.6	− 0.4
20	30.0	32.0	+ 2.0
21	47.0	43.8	− 3.2
22	41.0	38.3	− 2.7
23	35.0	36.0	+ 1.0
24	13.0	13.2	+ 0.2
25	22.0	22.6	+ 0.6
26	70.0	61.6	− 8.4
27	53.0	48.4	− 4.6
28	50.0	46.3	− 3.7
29	67.0	63.8	− 3.2
30	7.0	7.9	+ 0.9
31	8.0	6.5	− 1.5
32	3.0	3.0	
33	6.0	9.3	+ 3.3
34	17.0	19.7	+ 2.7
Averages	28.8 pct	26.8 pct	

Comparison of Thin Section Analysis of Different Column Samples

In Table 11 the data from the petrographic analysis of two column samples of Pittsburgh coal collected at the Bruceton experimental mine are shown. The coal in this mine when examined at a fresh face has the distinct appearance of uniformity from floor to roof. The petrographic analysis of the first column sample obtained from the mine verified the megascopic appearance of uniformity. To a certain degree, therefore, in view of known uniformity of the coal, the determinations of the petrographic components in these two column samples which are essentially the same represent a check on the reproducibility of the method of thin section analysis.

TABLE 11—*Petrographic Components in Two Column Samples of Pittsburgh Coal Taken 75 Ft Apart in the Same Mine, Per Cent*

Sample	Anthraxylon	Translucent Attritus	Opaque Attritus	Fusain
Column A	63	22	12	3
Column B	66	20	10	4
Components in column samples, Pittsburgh coal, from 5 mines				
Edenborn Mine Fayette Co., Pa.	50	44	2	4
Consolidation No. 63 Mine Marion Co., W. Va.	50	40	7	3
U. S. Experimental Alleg. Co., Pa.	63	22	12	3
Pittsburgh Terminal No. 9 Mine, Wash. Co., Pa.	65	24	7	4
Apex Mine Harrison Co., Ohio	57	36	6	1

In the lower part of the same table thin section analyses of five column samples of Pittsburgh coal from widely separated mines are given. These analyses show significant differences in the composition of the coal in terms of petrographic components but, of course, do not offer any proof of accuracy. If reliability of the technique of microscopic measurement is accepted as a demonstrated fact, the analyses of column samples obtained from variously located mines show that appreciable differences in composition can be quantitatively evaluated.

SIGNIFICANCE OF ACCURACY TESTS IN RELATION TO PRACTICAL APPLICATION OF THIN-SECTION ANALYSIS

There is no particular need to cite here the many lines of coal investigation in which coal petrography has proved to be useful, but it may serve the purpose of this report to close with a brief account of the role of petrography in coal hydrogenation.

Considerable research work has been done by the Bureau of Mines in testing various ranks and types of coal for hydrogenation yields. Furthermore, the various coal components and ingredients have been isolated in more or less pure form and subjected to the same tests. As a result of these tests it is now known that there is a definite relationship between the petrographic composition of a coal, ease of liquefaction, and yield of various products.[6] When the petrographic composition of a coal is known quantitatively, the reaction of the coal to liquefaction and the yield of liquid products can be predicted.

Anthraxylon, the chief component in all bright coals, will hydrogenate very readily and give an exceptionally high yield. Similarly all the ingredients of attritus that are conspicuously translucent, such as translucent humic matter, spores, pollen, cuticles, waxy-resinous matter, and oil algae, give high yields and are easily hydrogenated. There is also evidence that the translucent constituents of coal yield liquid products differing in quality. More phenols and aromatic hydrocarbons seem to be present, for instance, in the liquid product from anthraxylon than from that of spores, resins, and oil algae.

No practical way has been found to isolate pure, opaque attritus for special tests, but attrital coals having a very high content of opaque matter, such as some splints, give considerably lower yields and are much more difficult to liquefy than bright coals high in anthraxylon and translucent attritus. Experimental tests with high opaque coal fix the calculated yield in some tests as low as 40 pct for opaque attritus and a probable average of 60 pct yield for this component.

Fusain is the most difficult to liquefy and gives the lowest yield of any coal component. Yields of 5 to 25 pct have been obtained with megascopically "pure" fusain. It is believed, to account for the variation in yield, that a small amount of fusain which appears megascopically to be "pure" but when it is seen under the microscope actually is reddish translucent may liquefy more readily than most fusain which is completely opaque under the microscope.

A microscopic analysis reveals the heterogeneous nature of coal and furnishes reliable determinations of the quantity of various constituents present. The importance of this information in studying hydrogenation properties of coal is emphasized by the fact that coal constituents react differently and give varying yields when hydrogenated. It has been stated that the extent to which the liquefaction yield from coal hydrogenation tests can be correlated with and predicted from petrographic composition depends almost entirely on the accuracy and completeness of the petrographic analysis.

SUMMARY

The experimental work that has been rather briefly discussed in this paper furnished proof for the standard method of using thin sections and microscope equipped with micrometer eyepiece to determine the petrographic components in a coal.

The effect of progressive increase in magnification from low to high power in making microscopic measurements of opaque and translucent areas was tested by three operators using two microscopes of the same type but different makes, and it was found that reproducible results could be obtained by different operators, using microscopes of different make, within a magnification range of 40 to 712 power when opaque and translucent areas were distinct.

Areal measurements of opaque areas represented in two micrographs of coal thin sections were made with a polar planimeter. The results of these measurements with a manually operated instrument agreed very closely with microscopic determinations of the opaque attritus contained in the same areas of the thin sections photographed.

A study was made of microscopic measurements of opaque attritus, fusain, and anthraxylon, using series of thin sections prepared from a large block and column sample of coal and completely and partly transecting each thin section. The data from this work were analyzed and led to the conclusion that, to make an accurate microscopic determination of coal components by transecting a thin section with eyepiece micrometer, it is not necessary to scan the section completely. If a single transect is considered as an increment sample, the number of transects or increments spaced at equal intervals across the section need not exceed in number 20 pct of the possible number to reproduce the results of complete coverage. Also, in evaluating petrographic components in a large block or column sample, two spaced transects in each section of the series give results that essentially duplicate complete coverage. The importance of experimental evidence in confirming the accuracy of the standardized procedure used of sampling two transects in each section when making a petrographic analysis of a column sample is emphasized in the great saving of time in accomplishing a piece of work requiring painstaking application. The time required to do an analysis by sampling two transects in each section is less than 5 pct of that necessary to cover completely every section in the series.

It is realized that all problems peculiar to this method of coal petrography have not been completely solved. However, perhaps this work will instill confidence in the reliability and usefulness of thin section analysis of coal and stimulate further critical work in all phases of coal petrography.

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DISCUSSION

J. M. Schopf*—Much uncertainty has existed in the minds of many coal petrographers and coal technologists, not only as to the accuracy and reproducibility of results obtained through the Thiessen method of analysis, which Messrs. Parks and O'Donnell have discussed here, but also with regard to the actual procedure of making the determinations. Careful reading of the present paper is more revealing as to the real procedure followed for the last 18 years at the Bureau of Mines laboratory than any other paper that has yet appeared. It is difficult to account for some of the earlier statements purporting to bear on critical procedure, and it is perhaps pointless to do so. I personally believe misconception and inaccuracy in earlier reporting of procedural details was due, on the one hand, to an ill-conceived effort at over-simplification and popularization in what should have been purely technical reports and, on the other, to an over-awareness of theoretical implications, particularly botanical considerations, that are entirely subordinate to the actual procedures used in this quantitative system of analysis.

The point to be emphasized is the great importance of the present paper by Parks and O'Donnell for anyone who is concerned with the Thiessen system of coal petrography. While probably no one would suggest that this method is applicable to all the various problems of petrographic coal analysis, it is with little question the best method yet devised for accurate comparison of column or block specimens. A critical treatment of the method is therefore of great interest. The data presented in the paper have their most important bearing on the reproducibility and accuracy of analyses of individual coal thin sections. The further combination of the individual section analyses, through column or bed profile to represent the petrographic composition and lithologic combination of entities in complete beds of coal, still deserves further critical consideration. It is hoped the authors will present later this equally pertinent information about degree of accuracy obtained by various methods of combining thin section analyses for general column sampling and standards of accuracy requisite for the various problems, such as selective degradation and segregation of components, the reasons for differences in washability, and other coal beneficiation processes on which petrographic study throws particular light.

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Although the practices used by Parks and O'Donnell were reduced long since to standard procedure in the Bureau laboratory, much that is new appears here for the first time. A few items deserve special comment because of emphasis or minor difference of opinion.

Table 1 is called "Type classification," and in this table the critical thresholds of component concentration are presented, mostly for the first time. Only five "types" are recognized. I believe the Thiessen method should be used for differentiation of more types than this—that a greater number of type varieties of coal exist that are significant and worthy of denomination. It should be mentioned, incidentally, that the so-called "subtype" classification (an-at, at-an, and so on) that has been used by the Bureau but is not mentioned in this paper, is unsuitable for the purpose. The "an-at" subtypes were based on the same microscopic data as the "type" classification given here, but, instead of representing a more refined analysis of these data, simply involve its recombination on a more generalized and less significant basis. Probably the authors realize this now, and it was not as evident when the "an-at" subdivision was first proposed, but it should be mentioned since otherwise some confusion will remain on this account in the literature.

Attention also is directed to the new *quantitative* definitions of cannel and boghead coal proposed here in Table 1. Both of these are nonbanded varieties of coal, and the quantitative threshold of 5 pct or less anthraxylon is, in my opinion, a practical, necessary, and appropriate basis for this lithologic distinction. It has been in use in the Bureau of Mines Laboratory for a long time but has not been publicized except through personal communication. It can be regarded nevertheless as a long-tested basis of differentiation.

I am less confident of the other qualifications differentiating boghead and cannel. There are many cannels that have significant amounts of opaque attritus that, in part, fail to be included, if we assign a quantitative significance to the term "predominantly" (50 pct + ?). Likewise a "predominant" concentration of "oil algae" is a remarkably high requirement in the differentiation of boghead coals. If this means 50 pct concentration, it is questionable whether boghead coals of this nature exist in America. It also raises the objection that many of our nonbanded coals having oil algae *as the dominant waxy components* still will not be classifiable according to these criteria, simply because the algae compose less than 50 pct of the whole coal.

The whole subject of type classification of nonbanded coals and allied rocks is confused in the literature with free implication and expression of conflicting opinion if one chooses to look for it. However, humic and opaque matter are usually the dominating petrographic components, and the more volatile waxy constituents, which presumably influence the chemical properties to a notable degree, are often present in much more subordinant concentrations than would be supposed without specific and quantitative petrographic determination.

I believe these distinctions proposed by Parks and O'Donnell for boghead and cannel coal deserve reconsideration and, since the "predominant" qualifications offered are essentially new with this publication, the objection about disturbing a stabilized usage is not truly applicable.

It is hoped that publication of the microquantitative criteria for differentiation of bright coal, semisplint coal, and splint coal will clarify these matters of usage. It has not always been appreciated that "bright coal" was not differentiated on the basis of luster, according to these criteria most lignites will be classed as "bright coal" although they have a very dull or earthy appearance. It will be appreciated also that the terms splint and semisplint coal, being established solely on the basis of opaque attritus concentration, have no essential connection with the terms clarain and durain sometimes applied with reference to megascopic appearance. Many coals regarded as durains will be classed as splint or semisplint if analyzed microscopically, but microscopic study is the only way of determining whether or not this is true. The term "splint" may be nonspecific as to petrologic composition for as much as 70 pct of the mass of the coal, but the chief advantage in using the term in this sense is that it directs attention to what is possibly the most important single item in petrographic evaluation of coking or hydrogenation characteristics. Usually the nongeneralized specific results regarding other microcomponents have been given for both the splinty and nonsplinty portions of coal beds analyzed by the Bureau of Mines, so that the somewhat nonspecific, yet precise, usage of the term has caused no misconception for detailed analyses. A serious misconception arises only if one assumes that splint, as defined in this sense, is the equivalent of durain, which is defined on quite another and less precise basis.

A point that perhaps deserves comment but is omitted from this paper concerns the quantitative definition of anthraxylon. This component is identified at higher magnification, and so in general it may be the most accurately determined of any of the coal components. Parks and O'Donnell show that equally accurate results may be obtained for opaque attritus when special analytic attention is given, when the coal sections are of high quality, and when extraneous mineral matter is not present so that confusion can result.

According to standard practice of the Bureau of Mines laboratory, anthraxylon is not identified in any particles or strands thinner than about 0.014 mm. This value, or subsize threshold, was chosen empirically because the subsquares of the Whipple disc, in combination with the binocular microscope, and lenses, used at the Bureau of Mines for all these studies, was determined to be about 0.014 mm when calibrated in the standard fashion. No strands of anthraxylon have ever been identified in strands appearing thinner than the "height" of one of the Whipple disc subsquares (⅕ square) throughout the petrographic work at the Bureau, according to information given me by O'Donnell while I was closely connected with this work during the war. The microscope used in these studies was always the same, and I feel, in view of the large amount of work performed with this equipment in which the 0.014 mm boundary was a convenient one, that it, or some close approximation of it, should be regarded as part of the accepted definition for determining anthraxylon quantitatively. Only in this way can the distinction between anthraxylon and vitrain be brought out adequately.

It was always supposed that the eyepiece magnification used for anthraxylon determination was about 100 diam. It now appears from Table 2 of Parks and O'Donnell's paper that this was not exactly true and that, if it had been, the subsquare value should have been 0.022 mm instead of 0.014 mm. It is well known, of course, that manufacturers of optical equipment of this character prepare their lenses critically for their optical correction and definition and that stated magnifications are only approximate, the visual enlargement being a quite incidental consideration if resolution is not impaired within the general magnification range required.

It thus seems that Table 2 of Parks and O'Donnell's paper should be taken as a general guide, only on the assumption that the stated magnifications of microscope lenses are accurately given. The subsize threshold value for anthraxylon determination probably has never been exactly at 100×, however, and if a person is interested in duplicating analyses as they have been performed at the Bureau laboratory, the humic or vitrinized strands thicker than 0.014 mm, regardless of actual magnification, should be regarded as anthraxylon. Similar but smaller particles were regarded by Thiessen as translucent humic or cell-wall degradation matter and, as such, are constituents of the translucent attritus—a component not directly estimated but determined by difference.

The subsize threshold that has been used for determination of fusain petrographically is approximately as given for the 60× magnification in Table 2—viz., 0.037 mm. All fusinized material in smaller fragments has been assigned to opaque attritus according to usual procedure, as vouched for by Mr. O'Donnell, the only person who has been associated continuously and closely with these studies of quantitative coal petrography from their inception in about 1930.

In closing this commentary I think particular credit should be given to the authors for the presentation of the accuracy tests of analyses of coal thin sections. I believe no one who has long been associated with this work has questioned its reliability as an accurate basis for comparison of coal. For that reason, perhaps, there has not been as much of an incentive to establish the limits of accuracy with statistical definiteness as would have been the case if consistency in the method had been less obvious. However, this confidence may not always have been shared by those who could only read about the method since details were not provided that would permit its exact and independent duplication. Possibly largely for this reason the method has remained substantially endemic to the Bureau of Mines Laboratory at Pittsburgh, and the only other place I know of where similar quantitative results are being obtained is at the Geological Survey of the Union of South Africa, where the "know-how" was conveyed by personal instruction. The present paper should do much to dispel doubts about its validity and accuracy when appropriately used under favorable conditions.

I am convinced the method deserves far wider attention than it has heretofore received and that Parks and O'Donnell have done much to widen its circle of usefulness by the presentation of this paper.

B. C. PARKS and H. J. O'DONNELL (authors' reply)—The authors express sincerest thanks to Dr. Schopf for careful reading of the paper and furnishing the pertinent comments in the foregoing discussion.

In regard to boghead and cannel coals, the critical amounts of components delimiting these types as given in Table 1 should probably be considered tentative. Bearing in mind the heterogeneous and variable nature of coal, arbitrary grouping into types of the most practical application and value can be made only after a complete survey of all natural occurrences or at least thorough sampling. Experience with boghead and cannel coals has been limited at the Bureau of Mines laboratory because of the insignificant importance in the United States of these coals compared with the common banded types. Petrographic analyses that have been made in our laboratory of American cannel coals show translucent attritus content in excess of 50 pct of all coal material. Likewise thin sections of American boghead coals show the predominant ingredient composing the translucent attritus to be oil algae remains. Whether or not boghead and cannel coals exist in the United States or foreign countries with petrographic compositions not strictly compatible with the table given, we are unable to say at the present time.

As Dr. Schopf states, anthraxylon in thin sections has been arbitrarily defined to exceed 0.014 mm in width. This value was accepted as a convenient unit of measurement for separating banded anthraxylon from attrital translucent humic matter. The unit of measurement was originally obtained by calibrating a Whipple eyepiece micrometer with a stage micrometer using a well known foreign make microscope. The smallest horizontal division of the eyepiece micrometer was found to be approximately 0.014 mm wide. With the combination of lenses used, the resulting magnification was assumed to be 100 diam. With the combination of lenses assumed to give magnification of 40 diam, the same division of the eyepiece micrometer had a value of about 0.0375 mm. However, with the particular microscope in question, the rated magnification of the lenses is valid only when used with a monocular tube and with a tube length of 170 mm. The fact was inadvertently overlooked that using the binocular accessory supplied for this microscope increased the magnification power of the lens combination used approximately 50 pct.

We have gone rather thoroughly into the matter of calibrating eyepiece micrometers with both the monocular and binocular accessories using two modern American biological microscopes. A wide range of magnification was tested using various combinations of oculars and objectives. We are now prepared to say that using either of the two American microscopes with binocular accessory the value of the smallest division on the Whipple eyepiece disc used for the lower limit measurement of anthraxylon varies from 0.014 to 0.015 mm when 15× compensating eyepieces are used in combination with a 10× apochromatic objective lens. The magnification is very close to 150 diam with either monocular or binocular accessory. The variation from 0.014 to 0.015 mm and close approximation only to 150 magnification is due to lateral adjustment of the eyepieces on the binocular accessory made necessary with different operators unless interpupillary distances happen to coincide. The calibrated value of the same division of the Whipple disc when a combination of 15× compensating oculars and 4× objective is used with the American microscopes varies from 0.036 to 0.038 mm. The magnification is approximately 60 diam. The value 0.037 mm ±0.001 mm is probably the most convenient for the lower limit of measuring fusain. Particles of fusinite smaller than 0.037 mm can rarely be distinguished from opaque attritus and for practical purposes can be classed thusly.

Stability of the Atmosphere and Its Influence on Air Pollution*

By Henry F. Hebley,† Member AIME

(Cincinnati Meeting, October 1947)

Introduction

How often has the thoughtful observer pondered the apparent contradictions experienced in the weather?

One can take records of two days, one in the winter and one in the summer. The "degree days" indicate that space heating and domestic load were present in one case and not in the other. The Business Index may be about the same, indicating that the industrial activity of the area and its concomitant use of fuel were about the same. Thus the volume of the products of combustion released to the atmosphere varied only as related to the space heating demand.

Yet on one day in winter the atmosphere may be clear and bracing, while on a summer's day the air is thick and murky and the visibility sharply reduced. The situation may be reversed. What are some of the factors that contribute to the creation of these conditions?

These few notes have been prepared to indicate some of the influences that are exerted on the problem.

The Sources of Atmospheric Pollution

The industrial area of Pittsburgh, Pa., has been a dominant center for the production of iron and steel for over one hundred years. In fact, with the growth of the numerous auxiliary processes Allegheny County which contains Pittsburgh is probably the most industrialized county in the world.

In addition, the type of manufacture carried out in this area is predominantly dependent on thermal energy (heat). That energy is supplied from coke and coal produced in the vicinity of Pittsburgh

Indicative of the stupendous quantities of fuel that are employed in this industrial area's economy, the following (Table 1) taken from the U. S. Census (1940) may be of interest.

Table 1[5]—*Comparison of Industrial Centers Based on U. S. Census of Manufactures* (1939)

City	Cost of Fuel	Purchased Power	Tons of Coal Consumed per Man Employed in Industry
St. Louis	$10,726,820	$ 6,702,551	15.5
Pittsburgh	60,934,813	14,682,878	86.5
Youngstown	32,413,140	5,724,874	
Philadelphia	32,285,789	15,391,466	13.2

In tonnage consumed in the area, the figures are equally impressive:

	Annual Tons of Coal
Metallurgical Industries	35,000,000 to 40,000,000
Commercial Installations (Pgh.)	1,500,000
Domestic hand-fired) (Pgh.)	750,000

An influence that has never been fully appreciated by many individuals is the fact that no matter what adventures the gases of combustion experience as they pass through the various processes, ultimately the spent gases are discharged through chimneys to the atmosphere.

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* Paper presented at Joint Fuels Conference of Coal Division AIME and Fuels Division ASME. Manuscript received at the office of the Institute February 24, 1948. Issued as TP 2395 in Coal Technology, May 1948.

† Director of Research, Pittsburgh Coal Company, Pittsburgh, Pa.

Forming a portion of those gases of combustion are quantities of the oxides of sulphur and nitrogen, both of which incidentally are colorless gases.

THE TOPOGRAPHY OF PITTSBURGH AND ENVIRONS

The development of the Pittsburgh area as a center of heavy industry rests on a number of favorable circumstances:

1. Good coking coal, yielding a strong metallurgical coke low in sulphur content. 2. Adequate supplies of limestone for fluxing. 3. Economic water transportation arrangements exchanging ore from the iron ore fields and solid fuel from the coal fields. 4. Ample rail transportation for finished products to the centers of consumption. 5. Excellent water transportation for coal from the mines to the coke ovens. 6. Convenient sources of refractories for use in the furnaces employed by heavy industry.

These very advantages, however, are accompanied by difficulties of topography associated with the steep valleys of the Allegheny, the Youghiogheny, the Monongahela and the Ohio Rivers. These are natural conditions and create local climatic situations affecting air pollution which bear earnest consideration.

Such conditions have been augmented by human activities through the location of rail and river transportation practically at the river level, together with the concentration of industry within the confines of the narrow steep valleys.

It may be of interest to observe that many of the centers of heavy industry are similarly situated:

Youngstown, O., in the Mahoning Valley.

Johnstown, Pa., in the Stony Creek Valley.

Wheeling, W. Va., in the Ohio Valley.

Sheffield, Yorkshire, Eng., in the Don and Sheaf Valleys.

Liege, Belgium, in the Meuse Valley.

Incidentally, all of these cities have atmospheric pollution problems very similar to those associated with Pittsburgh, although Sheffield and Liege, because of their proximity to the ocean, experience a maritime climate compared to the continental type prevailing in the American cities, excepting those west of the Rockies.

A study of the contour map of the Pittsburgh industrial area under consideration will indicate that the Monongahela and its tributary the Youghiogheny flow in a more or less northerly direction until they reach the town of Duquesne when the course changes to roughly a northwesterly direction.

The Allegheny maintains roughly a southwesterly direction within the area. The Ohio, formed at the confluence of these two rivers, continues in a northwesterly direction until it passes out of the district.

It will be noted that the steep valleys form convenient conduits through which the atmospheric pollution may flow by air drainage from the numerous industrial centers situated in this Ohio River drainage area.

WIND AND AIR MOTION

In such broken terrain with steep hills and valleys running in many directions the local air movements in the lower layers are extremely complex. At the higher levels there are for the most part wind movements with a westerly component. These great westerly air movements are very steady. However, under such local interference of terrain, the disturbing influences are very great when compared to those encountered in the cities of the plains.

The air drainage system that is present in the Pittsburgh area is quite apparent to those who have taken pains to study the local air flows of the lower atmospheric layers as they are encountered in the Monongahela, Allegheny and East Ohio drainage area. No description of the deeply complicated air movements in such local terrain can excel that given by Mr. T. Morris Longstreth in his book "Knowing the Weather."[1] He states: "Below an altitude of 600 ft the wind proceeds in an unpictural turmoil. Friction with field and wood and hill forms an irregular drag upon its undermost layer. The overlayers stub their toes, stumble forward, fall down, turn somersaults, make mill-wheels, make spinners, fluctuate in velocity, rush and crowd and pick themselves up and go on. The vertical convulsions of wind are less rapid but just as intricate."

Superimposed on these local fluctuations are the steady air flows down the tributary valleys into the main valleys of the Monongahela, Allegheny and Ohio Rivers. This condition is known as the air drainage system previously mentioned and provided weather conditions are conducive, cooled air will flow into the system from as far afield as Morgantown, W. Va., on the Monongahela, New Kensington on the Allegheny or New Castle on the Beaver River.

When the sloping surfaces of the sides of valleys radiate their warmth to the universe on clear nights, the ground drops in temperature and the layers of air touching the ground are in turn cooled and begin to slip down into the valley and drain into the valley system.

Such air drainage movements manifest themselves as cool breezes issuing from the mouths of valleys where they join the course of the main rivers. In the area under consideration, even though fog and mist have accumulated in the main river valley reducing the visibility to zero, the captains navigating the river craft are well aware when they are abreast of the valley of a tributary because of the cool drafts.

Such winds are known as "katabatic" or downhill winds, and although they are comparatively faint and gentle in this area they have been known to reach a velocity of 200 miles per hr as they sweep down the snow covered slopes of the plateau of Greenland toward the sea.

Air movements of this nature which are quite pronounced pour out of the canyons of the Wasatch Mountains that surround the valley of the Great Salt Lake in which Salt Lake City is situated.

In passing, it should be recalled that Salt Lake City experiences rather heavy atmospheric pollution during anticyclonic conditions. Also the Salt Lake City industrial area, embracing the town of Murray, contains considerable heavy industry. It is, however, desirable to give closer consideration to the vertical air movements previously mentioned.

These vertical convulsions of air creating great upward columns of rising air must be felt during a flight in an airplane to be appreciated. In contrast, the sudden emergence during flight from the rising current into quieter air and the ensuing sudden drop are equally convincing to the passenger of the powerful vertical movements of the atmosphere.

Physics of Air Movements

In any study of the problem of atmospheric pollution the influences of the turbulence and stability of the air are of paramount importance. These conditions are related to the general air drainage movement previously mentioned; but some of the local factors and terrain exert effects that greatly modify the final results.

In a fundamental study of air movements it must be borne in mind that gases are far more free to move than liquids. In addition, gas when heated expands and thus its density is decreased.

Many examples of this fact may be recalled but possibly the draft of the common chimney is the most apt illustration. In this device the air for combustion is warmed, experiences a reduction in density and is pushed upward by more dense (heavier) air entering at the lowest place—generally under the grate of the furnace. Another homely example that depends on the same principle is the gravity warm air furnace used in the space heating of homes.

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[1] References are at the end of the paper.

Considering now the solar heating of the earth's surface as it is affected by absorption of heat or insolation, that surface, because of numerous influences, is very unequally heated. As a result, the layers of air adjacent to the earth's surface, which are heated by conduction from that surface, receive a very unequal amount of heat. Such unequal heating of the atmosphere touching the ground results in upward and downward convection currents of air between ground areas of different temperatures.

At this point it may not be amiss to introduce the conception of "adiabatic" changes in temperature. Turning once again to Longstreth and his book "Knowing the Weather" there will be found the following simple definition:

"The word "ADIABATIC" is applied in the science of Thermodynamics to a process during which no heat is communicated to or withdrawn from the body or system concerned. Adiabatic changes in temperature are those that occur only in consequence of compression or expansion accompanying an increase or a decrease of atmospheric pressure. Such changes are also described as dynamic heating and cooling."

The atmosphere, because it is a gas, is able to be compressed or expanded and the layers of air close to the earth's surface are compressed by the weight of the atmosphere above them.

In accordance with Boyle's law, any parcel of ascending air reaches a higher elevation and lower pressure and will expand. Such expansion is work done and the energy required to perform that work is derived from the heat contained in the air. This action results in the cooling of the parcel of air.

If, however, the air sinks to a lower elevation, it is compressed and is thereby warmed. These temperature changes, it should be noted, are accomplished with no addition or subtraction of heat from the parcel of air under consideration. They are adiabatic changes. Fig 1 is suggestive of the action.

In a dry atmosphere when a parcel of air ascends, the dynamic cooling caused by its expansion is practically uniform and falls at the rate of 1°C per 100 m of elevation (5.5°F per 1000 ft). This relationship is known as the dry adiabatic lapse rate.

It is well known, however, that the atmosphere has associated with it varying degrees of water vapor. If there is a goodly percentage of it present in the ascending air, the cooling that it experiences due to expansion may bring that air to 100 pct relative humidity or saturation. At that point some of the water vapor will condense and form liquid droplets. In so doing, the latent heat is released to the air thus retarding the rate of cooling.

If the parcel of air continues to rise, additional water vapor will condense, releasing further heat, and the air will be cooled at a "saturated adiabatic rate," also known as the retarded, wet or moist adiabatic rate.

This rate of cooling is far more complicated for a number of influencing factors, such as the pressure and temperature and whether the liquid accompanies the parcel of air on its journey or is left behind.

Under different conditions the moist adiabatic rate varies from approximately 0.4°C to nearly 1.0°C per 100 m, or 2.2°F per 1000 ft to nearly 5.5°F per 1000 ft.

If actual data are not available, it may be convenient to assume a moist adiabatic rate of 0.6°C per 100 m (3.2°F per 1000 ft).

The ability of the atmosphere to carry water vapor increases with a rise in temperature. Therefore, air, if it is descending, is enabled to carry more moisture as it is compressed. As a result, there will be no condensation. If there is liquid in the parcel of air some or all of it will be evaporated into water vapor; but in so doing, latent heat will be drawn from the air and the warming effect due to compression will be retarded by the latent heat requirements.

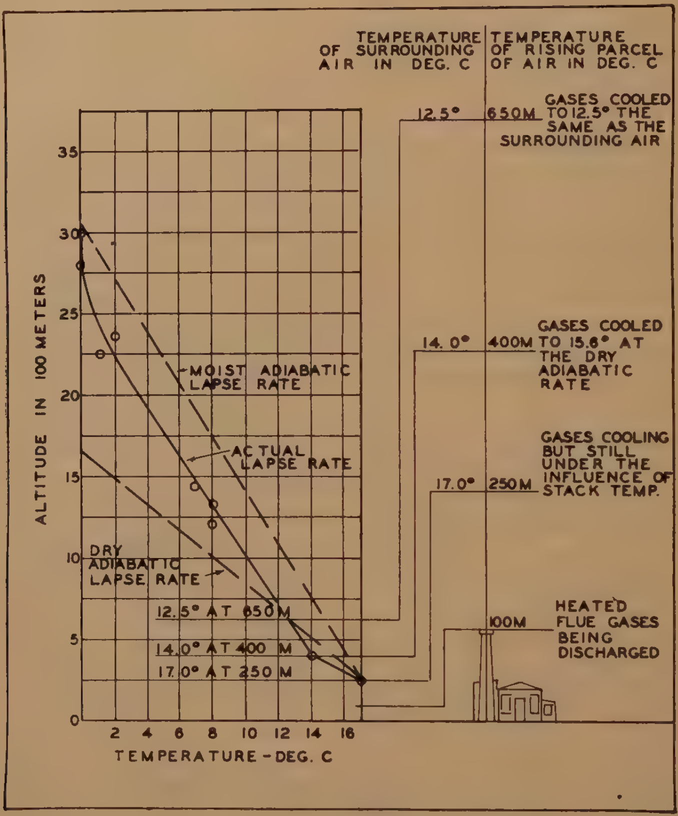

FIG 1—THE STABILITY OF AIR.

If all the water vapor contained in the ascending parcel of air continued to be associated with it and then descended again, the retardation of the rate of cooling would be the same as the retardation of the rate of warming during descent. However, if the moisture has been precipitated the parcel of air on its descent will increase in temperature at the "dry adiabatic rate."

Even though condensation has taken place, there may be some liquid present when the air begins to descend; but this changes to water vapor as the temperature rises and after that condition has been reached a continued descent of the air is on the dry adiabatic basis.

The discussion so far has dealt only with the changing temperatures of a parcel of air due to the varying pressures caused by altitude. However, many influences act frequently to cause the air to depart radically from the foregoing rule. Strong movements of horizontal air currents in the Pittsburgh area frequently sweep in from the Northwest or Southwest bringing cold or warm masses of air. In addition, air may not be ascending or falling. Consequently it will not be changing in accordance with the adiabatic rate. Also the atmosphere is continually gaining or losing thermal energy by conduction, absorption and radiation. Therefore, the various temperatures encountered at different elevations often show a marked deviation from the adiabatic rate.

In studying the change of temperature with height, it is referred to as the "lapse rate." It refers to the passing from a higher to a lower temperature. Where a stratum of warmer air is encountered and the temperature rises, the lapse rate is considered negative.

In actual experience, the atmosphere over a local area may have lapse rates which vary greatly between consecutive days. Also, there are variations in the lapse rates at different levels of the atmosphere during 24 hr. This condition is especially true for the first 11,000 ft from the surface of the ground. Above this elevation the lapse rates are likely to be more uniform.

It is in this 11,000 ft band of atmosphere that the temperature changes are very irregular and are modified by variable wind movements, gusts, eddies, and the like, caused by valleys, hills, bodies of water, and other topographical conditions. At times these local effects create a temporary lapse rate that is in excess of the dry adiabatic rate. Conversely, the lapse rate is frequently less than the moist adiabatic rate.

An actual increase in air temperature with increase in altitude is known as an Inversion of the Temperature Gradient, or simply an Inversion.

When such an inversion condition exists, the inability of the lower layers of the atmosphere to rise keeps them in a stable condition. After sunset on clear starlit nights with very gentle wind movement the surface of the ground cools rapidly by radiating its heat energy out to the universe.

The lowered ground temperature influences the adjacent 10 to 20 ft of air causing it to fall below the temperature of the higher air strata. Fogs known as radiation or ground fogs are frequently caused by this inversion condition provided other atmospheric factors are favorable to them. On the other hand, at the higher levels wind movements from different directions and having different temperatures may create an inversion.

As previously mentioned, a convenient assumption for the moist adiabatic lapse rate is 0.6°C per 100 m (3.2°F per 1000 ft). This rate is also close to the average existing lapse rate as determined by many observations in different countries.

The Stability of the Atmosphere

From what has been stated, it will be realized that air movements are influenced by many factors which act in an extremely complicated manner. These air movements can be studied through the relationship of the lapse rates to the tendency of the atmosphere to descend, ascend or stay in its position.

It may be profitable to consider these three cases in greater detail:

1. If a layer of air, say 1500 ft thick, rests on the ground and has practically the same temperature throughout that height, the air against the ground will be the most dense (heaviest in weight per cubic foot) because of the weight of the air column resting upon it. With no external influence this air will continue in the lower layers.

Now assume that some of this air is forced to ascend (possibly as chimney gases) through the surrounding air. It will begin to expand because of decreased pressure and, provided it does not reach the saturation point (RH 100 pct) it will cool at the adiabatic rate.

Thus, if it is still in the atmosphere of constant temperature, the parcel of air forced upward will be cooler and more dense than the air surrounding it. As a consequence, it will subside to its position at the surface. Such a condition may be suitably illustrated by atmospheric conditions that prevailed in Pittsburgh at 11:00 A.M., March 31, 1947 (Fig 2). In the analysis of the particular day under study, some attention should be given to conditions close to the ground. These are presented graphically in Fig 3. The air layer close to the ground surface, that is 100 m up, on that day had considerable turbulence. The panel of air (*c*) at 5°C if given an initial push, possibly from some increase in temperature, will start on its way upward cooling at the dry adiabatic lapse rate as it travels. After rising some 100 m the temperature of the parcel of air (*C*2) will have receded to 4°C. However, the surrounding atmosphere has a temperature of +2°C. Thus the air will continue to rise and "punch" into the layer of stable air resting above the turbulent layer.

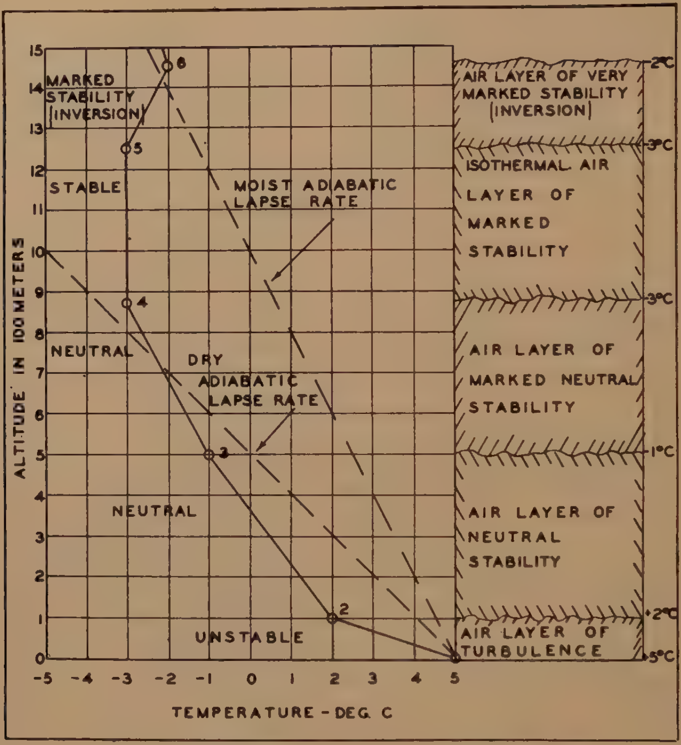

FIG 2—TEMPERATURE LAPSE RATE WITH ALTITUDE.

A portion of the lapse rate for the next layers of the atmosphere is shown by Fig 4. The line 1-2-3-4 is the actual plot of the air temperature at different altitudes.

It is of interest to ascertain the stability of the air.

The parcel of air (*A*) is moved up in elevation by one of a number of causes, say turbulence, and reaches the elevation (*A*2). Because of expansion due to decreased pressure, the parcel of air with a relative humidity of 46 pct has cooled at the dry adiabatic lapse rate and has been reduced in temperature to −0.6°C. The actual temperature of the atmosphere surrounding the parcel *A*2 is 0.0°C.

Thus the air parcel is cooler and therefore more dense and heavier than the surrounding air and if it experiences no further interference will sink back to its original position. (An atmospheric condition that creates this state is stable air.) In passing it should be noted that the air with a relative humidity of 46 pct was in the unsaturated condition.

Had the parcel of air been in a saturated condition, its action may have been modified. If the slope of the 2-3 portion of the lapse rate line 1-2-3-4 had been parallel to the moist adiabatic lapse rate (0.6°C per 100 m) then the parcel of air would have dropped in temperature at the moist lapse rate and the air would stay where it came to rest. In other words it would be neutral, as its changes would coincide with those of the moist lapse rate.

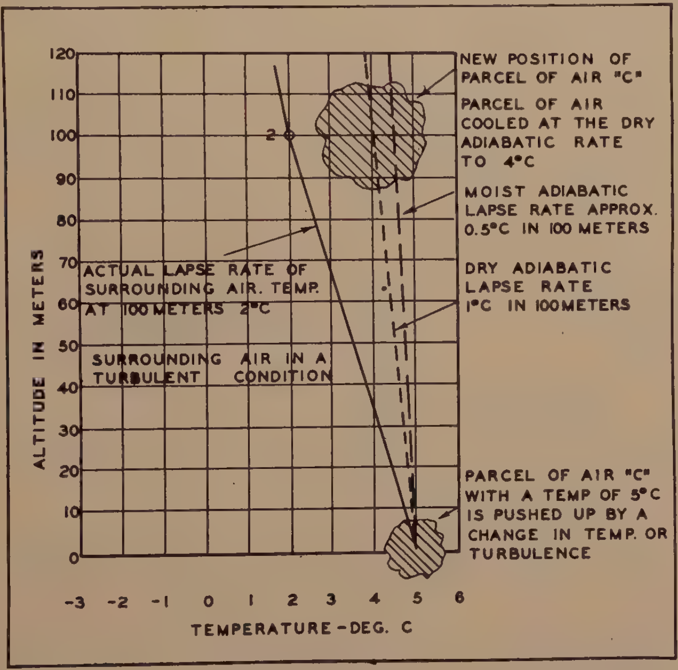
120
110
100
90
80
70
60
50
40
30
20
10
0
ALTITUDE IN METERS
-3
-2
-1
0
1
2
3
4
5
6
TEMPERATURE - DEG. C
2
NEW POSITION OF PARCEL OF AIR "C"
PARCEL OF AIR COOLED AT THE DRY ADIABATIC RATE TO 4°C
MOIST ADIABATIC LAPSE RATE APPROX. 0.5°C IN 100 METERS
DRY ADIABATIC LAPSE RATE 1°C IN 100METERS
ACTUAL LAPSE RATE OF SURROUNDING AIR. TEMP. AT 100 METERS 2°C
SURROUNDING AIR IN A TURBULENT CONDITION
PARCEL OF AIR "C" WITH A TEMP OF 5°C IS PUSHED UP BY A CHANGE IN TEMP. OR TURBULENCE

Fig 3—Atmospheric turbulence—a high lapse rate causes instability of air.

It so happens that in the higher layers a condition of marked stability known as an inversion of the normal lapse rate was encountered. (Fig 2, points 4-5-6-7). Here the air actually became warmer than the layers below.

The extremely powerful ability to imprison impurities associated with the products of combustion as they are discharged into an atmosphere of such stable conditions has not been fully realized. It needs greater consideration.

In this case the prevailing conditions are of such marked stability that even if the relative humidity were such that condensation took place, the temperature difference would still be sufficient to hold the atmospheric pollution within bounds.

Conditional Instability

So far in this discussion, the conditions of the atmosphere were such that the influence of the "dew point" has not been a factor in the cases of stable and unstable air. All that has been shown is the tendency of the air to return to its original state and position thus gathering the impurities created both by nature and man and holding them until atmospheric conditions dispel them.

On numerous occasions however the amount of water vapor associated with the air is a great deal more than the quantity encountered on March 31, 1947.

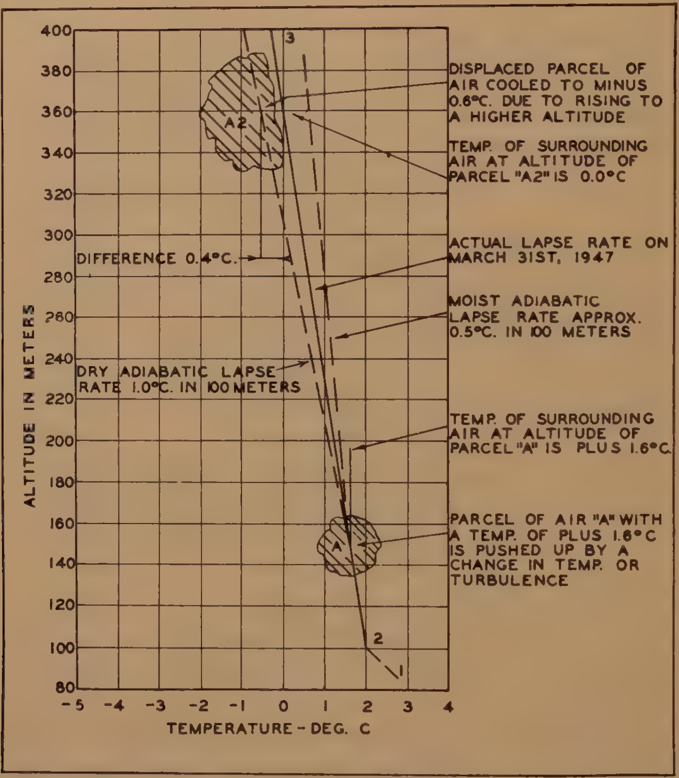

FIG 4—ATMOSPHERIC TURBULENCE—A LOWERED LAPSE RATE INCREASES STABILITY OF AIR.

The lapse rate will follow the dry adiabatic line until the dew point is reached. After this, the lapse rate will follow the moist adiabatic rate which is retarded compared to the dry adiabatic because of the continual release of latent heat as the water vapor condenses into drops of liquid.

The foregoing discussion pertaining to the influence of lapse rates on the various layers of air has taken little account of convection and turbulence. On warm sunny days when the surface of the ground rapidly becomes heated, the adjacent layers of air increase in temperature and begin to rise causing convection currents, thus aiding in local vertical air circulation that is inherent during unstable air conditions.

In addition to this movement of convection there is the vertical component of air velocity brought about by wind or air in motion. Such currents of air do not require that the atmosphere be unstable in character. As described in the previous quotation from Longstreth, the air layers adjacent to the ground, the hills, valleys, and obstacles such as buildings, trees, and so forth, cause local irregularities in the air movements. This results in an uneven reduction of the velocity of the ground layers and a downward sweep of the upper layers that are moving at a higher velocity. Such surges thrust the ground layers upward in an irregular fashion giving rise to gusts of wind both laterally and vertically and a combination of these movements in an indescribably complex manner.

Such turbulence will carry the surface air layers to higher strata causing appreciable mixing of the atmospheric impurities through a rather deep layer of the atmosphere and thus diluting the pollution by distributing it through a vastly greater volume. Turbulence drops rapidly as the stability of the atmosphere becomes more marked, and where low inversion conditions prevail it is practically nonexistent, especially at the ground level where friction exerts its influence.

The effectiveness of turbulence on the scavenging of the impurities from urban and industrial atmospheres must not be underestimated.

Civilization has, up to the present, depended on turbulence, convection currents, and air movement to cleanse the urban atmospheres of industrial cities. Most of the time atmospheric conditions perform that function rather successfully; but when the air movements fail, then the pollutants become concentrated with their disagreeable attributes.

The unprecedented growth of the world's industrial activities have aggravated the conditions, especially in cities whose growth has taken place through many decades with little or no city planning and with buildings concentrated in narrow streets interfering greatly with effective ventilation of the area. Thus the atmospheres of urban centers offer potential encouragement for foggy and smoggy conditions provided other atmospheric factors favor the fog formation.

Dobson[2] has brought attention to the subject when he pointed out that although boisterous winds of high velocity will sweep out atmospheric pollution, the factor of turbulence is a powerful one in dissipating the pollutants. It is possible to have fairly rapidly moving winds with but slight turbulence. Such air motions may not be as efficacious as gentler winds which are associated with strong upward and downward air movements.

Dobson[2] further indicated that often the atmosphere at 1000 ft is moving at a far greater velocity than near the ground level and is relatively free of pollution. If turbulence is present there is a ready mixing and dilution of the air impurities with this clean air layer that is just a few hundred feet above, rather than being swept through the city area from windward to leeward, a distance in many cases of over 30,000 ft.

This condition has been mentioned by Meetham[3] who points out that the concentration of impurities in the central area of an industrial city is more or less constant no matter how the wind changes in its direction. Fig 5 has been postulated for the Pittsburgh area based on Dr. Meetham's[3] analysis to indicate a possible distribution of atmospheric pollution when wind in its motions of translation moves the atmospheric pollution down wind. Naturally each industrial center is a problem unto itself and should be analyzed as such; and this diagram has been presented only to provoke thought on the problem.

Assuming the maximum concentration of impurity in the center of the city as represented by 100 arbitrary units of air pollution, it will be noted that it trails down wind in decreasing intensities, but is still carrying 25 units 4 miles from the center of the city.

To the windward, appreciable concentrations are encountered one mile from the 100 unit concentration. If concentrations of over 70 arbitrary units are considered as heavy, it will be noted that in the hypothetical case assumed there is an area in the center of this city bounded by a circle of one mile radius that will experience atmospheric pollution when the wind is blowing from any quarter. Such a postulate may be studied for the conditions of any industrial center as well as the proximity of other neighboring communities of industry.

In the case of Pittsburgh, the city is surrounded by industrial centers of 25,000 population or more. Fig 6 indicates the problem of transported air pollution derived from exterior sources.

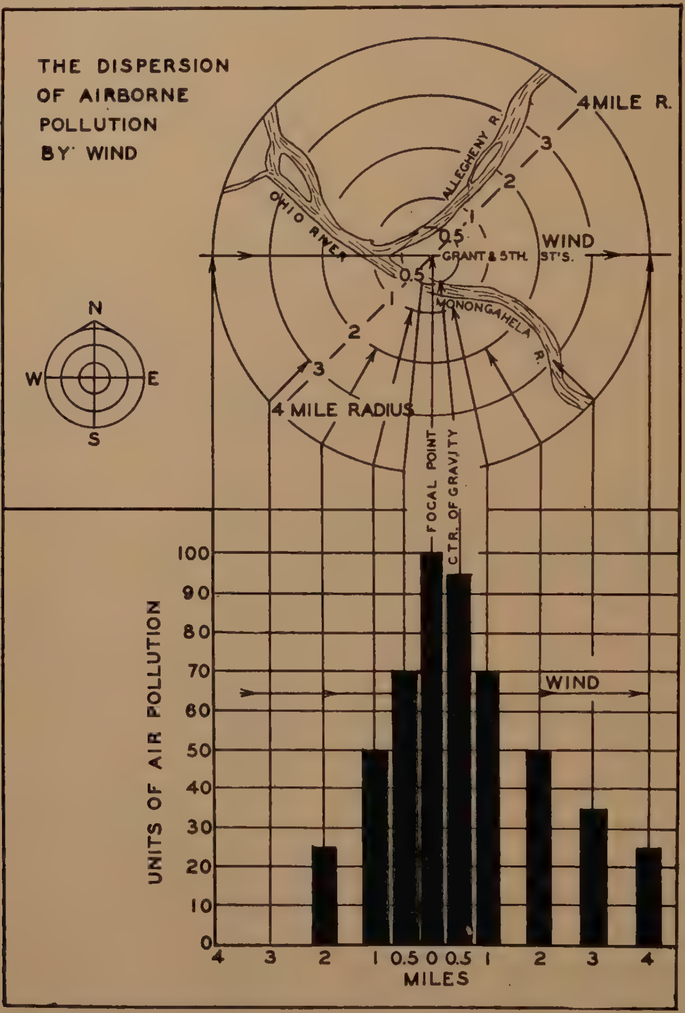

Fig 5—The dispersion of airborne pollution by wind.

In order to convey some idea of the relative effect of wind velocity and atmospheric turbulence, Dr. Meetham has offered a numerical demonstration and has graciously permitted the use of the salient points of his analysis in this text: "It is convenient to begin by imagining a row of cubical cells, in one of which there are 64 units of smoke. Diffusion due to turbulence may be represented by supposing that in one second's time the smoke is divided equally vertically between two adjoining cells."

Fig 7 shows the position of the smoke at successive time intervals of one second, due to turbulence alone. To take wind into account, suppose that a chimney emits 64 units of smoke each second into the cell (10 ft cube) immediately above it and that the wind velocity is 10 fps.

By an extension of Fig 7 it can be seen that the amount of smoke in different positions down wind from the chimney top is given by Fig 8.

It should be noted that no allowance has been made for horizontal diffusion and the figure best represents the effect of a row of chimneys placed across wind at right angles to the plane of the paper. A heavy line has been drawn enclosing all numbers greater than 15, which was selected arbitrarily. It represents the apparent outline of the smoke trail from a single chimney when it is viewed from one side by an observer who can detect 16 or more units of smoke in any line of sight.

In practice, smoke trails are never as steady as this because they develop waves due to large scale turbulence. Another effect of large scale turbulence is to cause a progressive increase in the rate of diffusion. Therefore Fig 8 must be regarded only as a simplified abstraction.

Fig 9 shows the effect of doubling the wind speed. Every 64 units of smoke at the chimney top now has to fill two cells and the smoke moves along at twice the speed. The visible outline of the smoke is much reduced in area.

Fig 10 shows the effect, with the original wind speed, of doubling the rate of diffusion due to turbulence. The width of the smoke trail increases rapidly at first and again the visible outline is much reduced in area. The results as presented in Fig 7 to 9 inclusive were based on the actual studies made at Liecester, England.

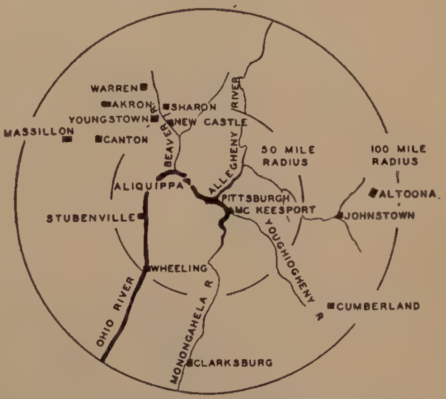

FIG 6—POTENTIAL SOURCES OF AIR POLLUTION SURROUNDING PITTSBURGH. (CITIES LARGER THAN 25,000 POP.)

There is an interesting result if the concentrations of smoke along the axis of the trail in Fig 7 to 9 inclusive are presented as in Table 2.

The figures shown in parenthesis were obtained by dividing the top row of numbers by the square root of 2.

TABLE 2

Fig	Wind Speed Fps	Diffusion Rate	Distance in Feet from Chimney Top								
			0	20	40	60	80	100	120	140	160
8	10	1	64	32	24	20	18	16	15	14	13
9	20	1	32		16		12		10		
			(45)	(22)	(17)	(14)	(13)	(11)	(10)	(10)	(9)
10	10	2	64	24	18	15	13	12	10	10	9

Their agreement apart from the irregularities at the chimney top with Fig 9 and 10 suggests that doubling the wind speed, or doubling the rate of diffusion due to turbulence, reduces the concentration of smoke in line down wind by a factor square root of 2 where the source of pollution is a row of chimneys in a line set across wind. If allowance is made for lateral diffusion across wind the result which is applicable to a single chimney is rather simpler—doubling the wind speed, or doubling the rate of diffusion, halves the axial concentration of smoke downwind from a single chimney.

It is also necessary to consider whether smoke is removed from surface air, either to the ground or to a region in the upper air from which it has a very much reduced chance of diffusing back to the surface. If the rate of removal up or down is large enough to be a controlling factor in determining the concentration of smoke in the surface air, one of two things may be expected to happen:

1. If removal up or down is a function of the distance travelled by smoke, then it

FIG 7

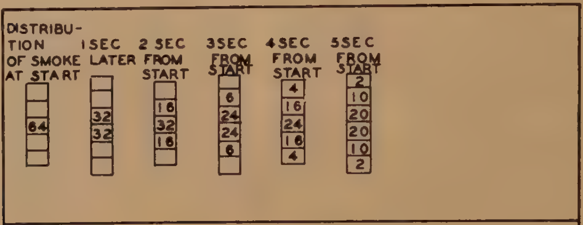

FIG 8

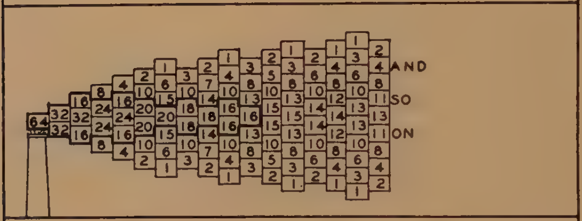

FIG 9

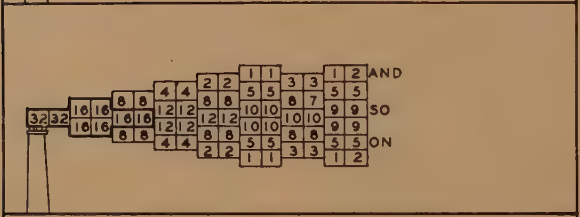

FIG 10

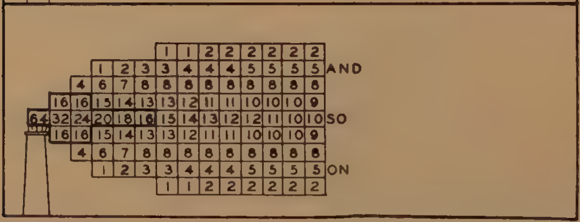

FIG 7—DIFFUSION OF SMOKE ALONG A VERTICAL LINE WITH NO WIND VELOCITY IN EFFECT.

FIG 8—DIFFUSION OF SMOKE ALONG A VERTICAL LINE WHEN INFLUENCED BY A WIND VELOCITY OF 10 FPS.

FIG 9—DIFFUSION OF SMOKE ALONG A VERTICAL LINE WHEN INFLUENCED BY A WIND VELOCITY OF 20 FPS.

FIG 10—DIFFUSION OF SMOKE ALONG A VERTICAL LINE WHEN INFLUENCED BY A WIND VELOCITY OF 10 FPS WITH DOUBLE THE TURBULENCE.

There is inadequate experimental knowledge regarding the pollution from a continuous source of smoke covering a large area such as a town. It is important to know first of all the concentration of smoke due to chimneys several miles away even as much as an hour after the smoke has been emitted.

should be expected that wind speed is important in determining the concentration of smoke in cities.

2. If removal up or down is a function of time only, then in a sufficiently large city wind speed will be unimportant and turbulence will be the dominating factor.

Both wind velocity and turbulence are extremely difficult to measure and little investigation has been carried out. However, based on the little knowledge that was gained during the Leicester, England, survey regarding the effect of turbulence gauged by the lapse rate, the measurements of daily average concentration of smoke were found to be consistent with following conclusions:

1. In any given wind speed, the concentration of smoke in central Leicester decreased linearly with the lapse rate.

2. In any given lapse rate the concentration of smoke in central Leicester varied at a rate proportional to the inverse square root of the wind speed.

In regard to the second conclusion, in any given lapse rate the turbulence increases with increasing wind speed, particularly in cities when the obstruction of the wind by high buildings produces large eddies.

This conclusion tends to indicate that in given degree of turbulence, the concentration of smoke may be almost independent of wind speed.

This condition in turn suggests that the rate of upward and downward removal of smoke from the surface air is important in a city. In addition, the rate of removal up or down depends on time rather than the distance travelled by the air. There is urgent need for more experimental work to be carried out on this phase of the problem; but based on the rather meager evidence that is available, the concentration of smoke in a city is much more strongly affected by turbulence than by wind speed.

The importance of the influence of turbulence on atmospheric pollution was fully realized by Dr. E. W. Hewson[4] and his associates during their study as an International Commission of the causes and remedy of the destructive effects of the flue gases discharged from the chimney of the smelter at Trail, B.C., Canada. This plant is situated in the deep valley of the Columbia River about 7 miles from the boundary between U.S.A. and Canada.

At times, under certain weather conditions, the destructive fumes of sulphur dioxide sweep down the valley to destroy crops and vegetation.

When inversions or isothermal conditions are experienced, the gases are effectively held within the steep confines of the valley and thus flow downstream, in what may be considered as a huge conduit. Samples of the atmosphere were taken at various heights above mean sea level and the sulphur dioxide concentrations in PPM were ascertained. The results of one series of determinations are shown in Fig 11 where the samples were taken during the night of May 17, 1940, from 1:35 AM to 1:50 AM. As Dr Hewson[4] points out, the horizontal stratification of the gas is quite marked. Such a condition was to be expected in the early morning when isothermal and inversion conditions existed.

Based on a mathematical statistical analysis of numerous smoke observations, Hewson and his associates found that the smoke during the early morning followed a pattern similar to that sketched in Fig 12. The stacks are approximately 400 ft in height, with the chimneys discharging at 2000 ft. The gases on discharge are about 100°F warmer than the surrounding air and with light wind the gases mix and drop in temperature at a greater lapse rate than either the dry or moist adiabatic lapse rates.

Upon reaching 2500 ft the gases reach equilibrium with the surrounding air and afterwards flow downstream as a ribbon about 200 ft in thickness and at nearly constant height.

As Hewson[4] has pointed out in the report of the investigations carried out at Trail, B.C., the heating of the sides of the valley is the factor that causes a cross movement of air in the valley from one side to the other. As the sun is rising, the westerly bank of a river running approximately N and S (for instance, the Monongahela) is heated more than the easterly side. As a result, convection currents lift parcels of air upwards along the westerly bank and there is a downward movement on the cool shaded eastern side. As the sun moves through the heavens, there comes a period during hours flanking noon when the action is more of less uniform throughout the valley, while in the late afternoon the easterly bank is warmed and the air movements are upward on that side. Because of the greater heights and the more rugged terrain of the Columbia River Valley these air currents are in all probability more

FIG 11

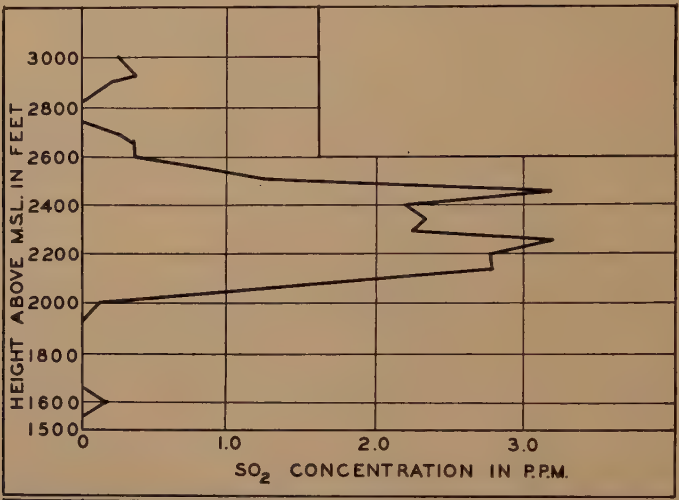

FIG 12

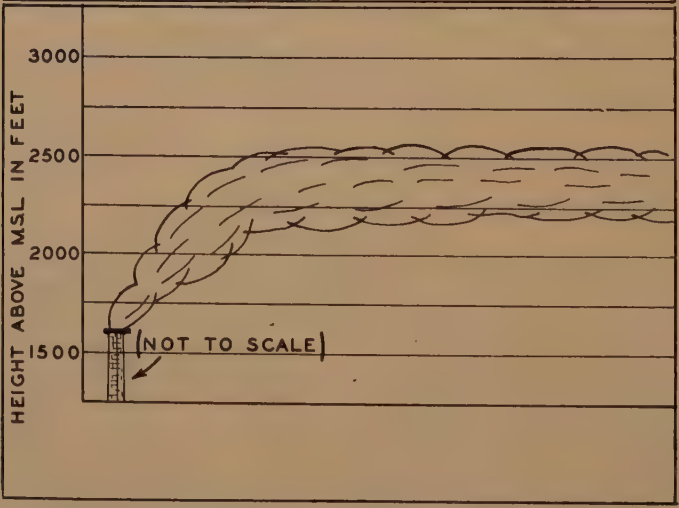

FIG 11—VARIATION OF SO_2 CONCENTRATION WITH HEIGHT.
FIG 12—BEHAVIOR OF SMOKE FROM STACKS WITH LIGHT WINDS AND TURBULENCE.

pronounced than those experienced in the Monongahela River valley; but this potential influence is present in most valleys.

From the foregoing statements it will be realized that there can exist a daily cycle of air conditions over a highly industrialized area, in which topography, location, and seasons combine to create polluted atmospheric conditions provided general large scale weather phenomena favor it.

CONCLUSION

No suggestions or methods are advanced as a palliative or cure for pollution that becomes concentrated in the atmosphere when stable air conditions as described are experienced. The problem is one that requires deep study and if this paper has in some small degree drawn thoughtful attention to a few of the factors which play such an important part in the problem of atmospheric pollution it will have succeeded in its purpose.

ACKNOWLEDGMENT

Grateful acknowledgment is given to Dr. A. R. Meetham of the Dept. of Scientific and Industrial Research in London for his helpful data, to Dr. A. W. Hewson of the Canadian Meteorological Service for his permission to use some of the data collected during the Trail air pollution investigation, and to the U. S. Government forecasters at Allegheny Air Port Weather Station for their cooperation in furnishing the lapse rates experienced in the Pittsburgh industrial area on certain days.

Lastly, credit is due to Mr. Jack MacLachlan of the author's staff for his patient preparation of the diagrams.

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Significance of Condensation Nuclei in Atmospheric Pollution

By Hans Neuberger†

(Cincinnati Meeting, October 1947)

INTRODUCTION

In our everyday life, visual perception plays perhaps a more important role than any of the other senses with which we are equipped by nature. It is, therefore, quite understandable that the average person associates with the term "atmospheric pollution," clouds of dust, palls of smoke or deposits of soot, phenomena which are so familiar to the residents of industrial cities. On the other hand, "air" is usually considered the prototype of the invisible, of the intangible, and any suspensions in the air which do not conform to this concept, are regarded as foreign to the atmosphere and thus as "pollution." According to Webster, the term "pollution" implies uncleanness, impurity, in short, something undesirable.

Although the scientist often accepts popular concepts, he is generally inclined to suspect that apparently "obvious" phenomena imply, literally, "more than meets the eye." To show that this suspicion is also justified with regard to atmospheric pollution, is the objective of this discussion. It will be shown that what seems foreign to the air at first sight, is by nature an integral part of it, and that the more important aspects of pollution generally escape attention.

THE ATMOSPHERIC CONSTITUENTS

We can distinguish two classes of atmospheric constituents: the permanent gases, chiefly nitrogen, oxygen, traces of argon, neon, and others, and the variable constituents which include water in its different states, carbon dioxide, ozone, and various suspensions, either in gaseous, liquid or solid form. These suspensions, known by the collective term "Aerosol,"[20] will be considered in detail.

DUST

There are two general types of aerosols: the first type is represented by the socalled "neutral" particles,[3] commonly known as dust and experienced by everyone, as it is the more conspicuous type of suspensions. It often represents a special problem to the mining engineer, the mechanical engineer, the industrial engineer, and many other professional men. Dust consists of finely divided solids derived from weathering, drilling, handling and preparation of rocks and coal. Dust may also consist of organic or inorganic particles resulting from manufacture of a variety of products such as flour, rubber, lime and steel, from traffic on land be it on the highways, on rails or in city streets, from decomposed vegetation or from volcanic eruptions. Perhaps the most reviled member of this class of suspensions is the soot and fly ashes from coal fires and other combustion processes.

The cosmic source of dust, namely the injection of meteoric matter into the atmosphere, is of minor significance because the concentration of particles is exceedingly small despite the relatively large mass of material which invades the earth's atmosphere from space.

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[20] References are at the end of the paper.

In general it can be stated that the atmosphere would contain dust even in the absence of the sources occasioned by human activities, because the chief sources of dust are *natural* ones,[4] such as volcanic eruptions and, particularly, dust storms originating in desert areas. It was estimated that the volume of fine ashes thrown into the atmosphere by a strong volcanic eruption amounts to as much as 100 billion cubic yards.[4] As another example of nature's prolific dust sources, the single dustfall of March 1901[9] may be cited, when about 2 million tons of dust from the African desert were deposited over Europe alone, not counting what remained in suspension and drifted with the air currents beyond the limits of the continent or what was deposited over the Mediterranean Ocean and elsewhere. In February 1903, 10 million tons of red dust from Northwest Africa were deposited over England.[19]

The size of the neutral dust particles ranges from about 3×10^{-5} to 10^{-3} cm and larger,[8] that is, from microscopic diameters to kernels visible to the naked eye. The median and modal size for large American cities was determined at about 5×10^{-5} cm.[10] It is not astonishing that it is particularly the large dust particles that arouse the ire of the housewife and other people concerned with the nuisance of dust. However, it should be noted that the large particles are in general rather shortlived as suspensions because they settle quite rapidly out of the air near their source, and once they are deposited, they cannot properly be called suspensions any longer, even though they may remain a nuisance.

CONDENSATION NUCLEI

The second type of suspensions which is by far more important than the neutral dust, is the "active" dust[16] which is made up of hygroscopic substances. Due to their affinity for water, the hygroscopic particles play an all-important role in the transformation of water vapor into liquid water droplets or solid ice particles. When the moisture content of air which is devoid of any suspensions, is increased to and above the saturation point, spontaneous condensation into water droplets will not occur until a supersaturation of roughly 800 pct is reached.[15] The fact that water vapor ordinarily requires some surfaces on which to condense under conditions of saturation or only slight supersaturation, is easily observed when drawing hot water in a bathroom. A deposit of dew droplets on walls, mirrors, windows, and other places, can usually be seen before a steam cloud will form throughout the room, if indeed it forms at all. In the free atmosphere, the hygroscopic suspensions furnish the surfaces necessary for the process of condensation, and therefore these particles are called *condensation nuclei*. As such, they are of vital importance in the formation of fog, clouds and, eventually, rain. The significance of rain in the maintenance of organic life on this earth is too well known to require further discussion.

The Chemical Nature of Condensation Nuclei—There is a great variety of substances which serve as nuclei due to their hygroscopicity. Chloride salts and sulphuric acid are perhaps the most important chemicals in the air acting as nuclei.[1,3,15] Other nuclei-forming substances are phosphorous compounds, oxides of nitrogen, and nitrous acid which may form from the nitrogen, oxygen and water vapor in the air either in the presence of ozone,[2,18] by lightning,[18] or by contact with glowing metal surfaces.[5] The importance of combustion processes for the production of nitrous acid was stressed by Coste and Wright.[5]

The importance of sulphur as a nuclei-producing agent was clearly shown by Ives and collaborators[10] who demonstrated the parallelism of the sulphur content of air and the number of condensation nuclei (Table 1) in several American cities.

TABLE 1—*Sulphur Content (in μg per m^3) and Average Number of Nuclei Per mm^3*
(After Ives et al.)

SULPHUR	NUCLEI
0–50	150
51–100	217
101–150	231
>150	316

The specific role of sulphur dioxide in the production of nuclei was investigated by Aitken.[1] He reasoned, and demonstrated by experiment, that sulphur dioxide under the influence of ultraviolet radiation is oxidized to the highly hygroscopic sulphur trioxide, which together with water vapor attracted from the air, forms minute droplets of sulphuric acid. He thus explained the frequently observable fact that in industrial areas a thin haze or smoke layer will often turn into a dense fog when the morning sun shines on it. However, a direct correlation between the intensity of the ultraviolet portion of sunlight and the number of nuclei at ground level is difficult to establish, because the ultraviolet radiation is not only affected by the property of the air layers at greater altitudes, but is also depleted by the nuclei themselves. Both effects were shown by Ives and Gill[11] who found that under clear skies (with strong ultraviolet radiation) the nuclei content was somewhat higher than under cloudy skies. On the other hand, they also demonstrated that more intense ultraviolet radiation was associated with a smaller nuclei content of the air, as is evident from Table 2, reproduced from the paper cited.

TABLE 2—*Average Ultraviolet* (UV) *Radiation* (*in Microwatts per cm^2*) *and Number of Condensation Nuclei* (N) (*Per mm^3*) *for Clear and Cloudy Skies*
(After J. E. Ives and W. A. Gill)

Sky Condition	Most smoky cities		Least smoky cities	
	UV	N	UV	N
Clear	24.2	240	36.4	172
Cloudy	8.1	207	16.9	162

This latter effect was also confirmed by H. Landsberg[14] at State College, Pa., a place remote from industrial activities, where a definite decrease of the biologically effective ultraviolet light was observed with an increase in number of condensation nuclei. That the observed effects are a function of the chemical nature of the nuclei involved, can be deduced from the fact that on the island of Sylt in the North Sea, the author[17] could observe the same effect of sky condition (as that shown in Table 2) only when the wind blew from the nearby mainland, while the reverse was true for a wind from the sea, as shown in Table 3.

TABLE 3—*Average Number of Condensation Nuclei Per mm^3 for Clear and Cloudy Skies*
(After H. Neuberger)

Sky condition	Wind Blowing from	
	Land	Sea
Clear	3.4	1.1
Cloudy	3.0	1.2

The winds from the land, aside from their higher nuclei content, apparently carry substances (most likely sulphur dioxide) from combustion processes which become nucleogenic under the influence of sunlight, whereas the nuclei in winds from the sea consist chiefly of chlorides and other ocean salts which are hygroscopic without a chemical transformation due to sunlight.

The transformation of sulphur dioxide to sulphur trioxide may also be caused by the presence of other oxidizing agents in the air, such as nitrous acid.[5]

It may be mentioned, that when the atmosphere is greatly supersaturated with water vapor, a condition which is rarely, if ever, found in nature, nonhygroscopic particles of very small size[12] may also act as nuclei. Similarly, positively or negatively charged ions will serve as nuclei in highly supersaturated air.[15] Such ions may be produced by radioactive material, cosmic rays, lightning, X rays, ultraviolet radiation, and others.[12]

The Sources of Nuclei

The chief natural sources of nuclei are volcanic eruptions, spray from ocean surfaces,[16] combustion products from forest and steppe fires,[4] swamp gases, exhalations from living or decaying vegetation, and bacteria. Man-made sources can be enumerated as follows: Industrial furnaces and processes of various kinds, such as blast furnaces, Bessemer converters, rolling mills, coke ovens, in short all industrial chimneys. Our modern transportation facilities, such as steam and Diesel locomotives and boats, automobiles and airplanes are also copious producers of nuclei. Domestic sources are oil-, gas-, or coal-fired furnaces, cooking stoves, incinerators, and tobacco smoke. In other words, nuclei are produced wherever substances or processes give off odiferous fumes,[2,16] where materials are glowing hot, or combustion takes place, *whether visible smoke is emitted or not.* Again, as in the case of dust, man-made sources are insignificant by comparison with natural sources.[4]

It may be in order to quote some numerical values in order to illustrate the variation in the nuclei content of air as well as the effectiveness of various sources.

An average grass fire extending over one acre, produces some 20,000 billion-billion ($= 2 \times 10^{22}$) nuclei. If these nuclei were distributed through a column of air having a cross-section of one acre and a height of 10,000 ft, there would still be a concentration of roughly 2 billion particles per cm^3. One puff of cigarette smoke contains some four billion nuclei according to Aitken.* Melander[16] found that the number of nuclei increased from about 2600 to 35,000 per cm^3 when the wind carried the smoke column of Mount Vesuvius over the place of observation at a distance of 6 miles. Measurements of J. H. Coste and H. L. Wright[5] showed that a flame of commercial coal gas lit for only 15 sec increased the number of nuclei in a chamber from 109,000 to 860,000 per cm^3. These authors also showed that the greater the sulphur content of the fuel, the more nuclei are produced by its combustion. Junge[12] computed that a flame of commercial coal gas which consumes 0.43 cm^3 gas per sec and contains only 0.003 pct sulphur dioxide derived from combustion of 0.073 mg of carbon disulphide in that volume, produces every second 250 billion ($= 2.5 \times 10^{11}$) sulphuric-acid particles of 7.5×10^{-7} cm radius. Salles* measured in the streets of Paris 40,000 nuclei per cm^3 which number increased to more than 100,000 when an automobile passed nearby. Amelung and Landsberg[2] found in a ventilated kitchen with a large gas range in use, the number of nuclei exceeding 500,000 per cm^3, while the outside air contained only about 25,000 nuclei per cm.[3]

The Physical Properties of Nuclei

As regards the size of the nuclei, it is, for average humidity conditions, of the order of 5×10^{-6} cm[3,5] and ranges from about 10^{-7} to 10^{-5}cm radius.[15] Their weight has been estimated at 10^{-14} to 10^{-18} g or an equivalent of an aggregate of 10^6 molecules. The minuteness of these nuclei is perhaps more comprehensively illustrated by the fact that 100,000 nuclei of spherical shape fill a volume of only about 5×10^{-11} cm^3, or that it takes 2 million-billion to occupy the space of one cubic centimeter. Thus, it is evident that, ordinarily, the nuclei are invisible. However, since their size is a function of the moisture content of the air, due to their hygroscopic nature, their presence may become apparent indirectly as haze, or as fog or cloud droplets,† under the proper moisture conditions. Experimental studies by Chr. Junge,[12] in good agreement with the theoretical analysis by H. Köhler,[13] revealed that the size of the condensation nuclei is practically unaffected by the humidity in the air as long as the relative humidity remains below 70 pct. Under this condition, the nuclei are either, solid, liquid, or gaseous particles, depending on their chemical composition. When the relative humidity increases above 70 pct, these particles grow slowly at first and then rapidly as the relative humidity approaches 100 pct. This is shown in Fig 1 by the solid curve.

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* Quoted from Landsberg.[15]

* Quoted from Landsberg.[15]

† The fact that a fairly large droplet condenses around each nucleus under condition of saturation or slight supersaturation, is utilized in the method of counting the number of nuclei in a "cloud chamber."

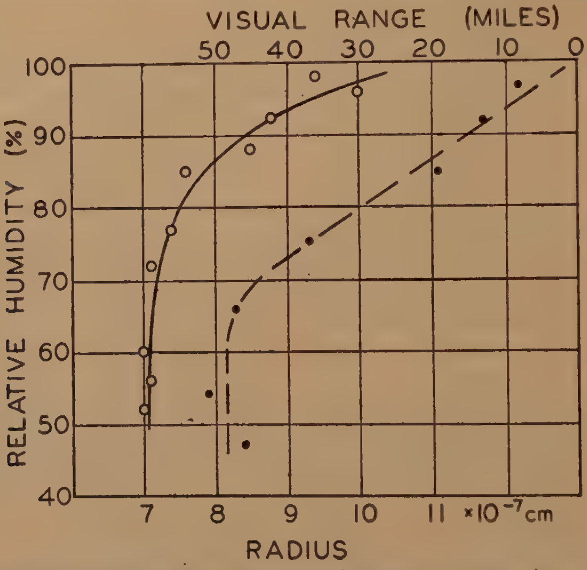

Fig 1—The radius of a condensation nucleus (after Chr. Junge) and average visual range as a function of relative humidity.

For comparison, the average visual range in miles was computed as a function of relative humidity from the material obtained by the author[17] on the island of Sylt, and plotted as a dashed curve. Since, at the location mentioned, the nuclear climate with respect to number and type is fairly uniform, it is to be expected that the visual range decreases with increase in size of the nuclei because the amount of veiling light scattered by the nuclei is a function of their size. This is well brought out by the portion of the dashed curve above 70 pct relative humidity. In general, the absorption of water vapor from the air by the nuclei at relative humidities above 70 pct transforms the nuclei from gaseous, liquid, or solid particles into tiny droplets of salt or acid solutions whose concentration varies with the original size of each nucleus and the amount of water absorbed. In turn, the amount of water condensed on the particle is a function of this concentration and the size of the particle. In addition, the presence of electrical charges on the nucleus has an influence on its hygroscopicity or, more precisely, on the vapor pressure. The various effects of the properties of nuclei on the condensation process can be summarized as follows: Water vapor will more readily condense on a nucleus, the greater its concentration of salts or acids, the greater its electrical charge, and the greater its radius. Thus, (neglecting the small effect of electrical charges) during the process of condensation, the growth of the smaller nuclei is slower than that of the larger ones, so that even dense fogs or clouds in relatively pure air contain a large number of free nuclei[15] which have not grown to the size of fog or cloud droplets, that is to a size of the order of roughly 10^{-3} cm^3. Only about 300 to 500 nuclei per cm^3 are needed for the formation of dense clouds.[15]

Fig 1 also shows quite strikingly that condensation is a continuous process that, in effect, begins at humidities far below the saturation point as determined for plane, pure water surfaces. Therefore, the decrease in transparency of the air with increase in humidity is also continuous, and the transition from clear air to haze to fog is so gradual that it is impossible to determine the instant at which the haze stops and the fog begins.

The visibility and stability of the condensation products is dependent on, among other factors, the size and the number of droplets. In industrial regions with their overabundance of nuclei, the transparency of the air is markedly reduced, when the relative humidity has not yet reached 100 pct. By contrast, in relatively pure country air, saturation may exist for some time before the fog becomes visible. Nevertheless, the meteorological conditions that lead to the saturation of the air near the ground, usually affect regions on such a large scale that the frequency of foggy days cannot readily be taken as an index of air pollution. This is evident from the fact that the average annual number of days with dense fogs for the period 1936–1940 inclusive, is 12 at Cincinnati, O., 11 at St. Louis, Mo., and 13 at State College, Pa. This means that the frequency of dense fog is practically identical at all three localities. However, if records of the duration of fogs were available for comparison, it is almost certain that the number of hours with dense fog would be the smallest at State College.

Comparison between the Two Types of Aerosol

Aside from the all-important role that nuclei play in the atmospheric condensation processes, they also surpass dust in point of number, greater persistence as an aerosol, and their chemical and, possibly, biological action.

As regards the concentration of particles in a unit volume of fairly clean air, the nuclei are about 50 times as numerous as the dust particles. For example, Giner and Hess[8] found in Innsbruck, Austria, the mean annual number of nuclei to be 24,100 per cm^3, the number of nonhygroscopic dust particles to be 464 per cm^3. In industrial cities,[10] the dust particles represent only a fraction of one per cent of the total amount of suspensions found in the air, as can be seen from Table 4. This table also shows that a certain parallelism exists between the concentration of dust and that of nuclei.

TABLE 4—*Mean Number of Condensation Nuclei* (N) *for Various Ranges of Dust Concentration* (D) *in City Air*
(After Ives and Collaborators)

D PER CM^3	N PER CM^3
<500	189,000
500–999	211,000
>999	223,000

This is not surprising since both types of aerosol are produced as a result of the same human activities.[10,15] A pronounced parallelism can also be found with respect to diurnal and annual variations of the atmospheric dust and nuclei content.[8]

In the free atmosphere, the nuclei exceed the dust particles by a factor of several thousand.[6] This is chiefly due to the fact that the dust, by virtue of its larger particles, is subject to considerable sedimentation,[20] while the smaller nuclei are easily carried to higher altitudes by light convection currents.[6,8] Measurements made in free balloon ascents,[6] showed for the first 2000 meters, an increase of about 25 pct in the ratio of number of nuclei to that of dust particles. Nevertheless, the nuclei content decreases with increase in altitude above the surface, even though at a slower rate, than does the dust content.[6,8,15] This decrease is due to dispersion and coagulation.[12] Similarly, the horizontal transport by wind generally carries nuclei to greater distances from their source than dust particles. The fact that salt nuclei from the oceans are found in the middle of large continents, illustrates the ease with which these small suspensions ride the air current. The previously mentioned cases in which dust from desert areas travels over great distances depend on special meteorological conditions, which are the exceptions, rather than the rule. At any rate, dust from industrial sources is usually deposited within a relatively confined area around the source, whereas the simultaneously produced nuclei are often carried far to the leeward.

As far as the cleansing effect of rain or of other forms of precipitation on the aerosol is concerned, it seems logical to assume that the large dust particles are more easily "washed" out of the air than the small nuclei. However, the results of pertinent investigations by various authors are by no means in agreement. For instance, according to the findings of Giner and Hess[8] at Innsbruck, rain decreases the number of nuclei "as a rule," while the dust content is "always" diminished. By contrast, Ives and collaborators[10] could not detect any effect even of heavy rain on the dust content of the air in American industrial cities. It is not possible at this time, to decide whether this discrepancy is due to the difference in the aerosol climate between the localities cited, or whether it is due to the difference in methods employed for the dust determinations.

Similarly inconsistent results become apparent when investigating the effect of precipitation on the number of nuclei. Not only does the nuclei content often not diminish during or after rain, but also in many instances, a pronounced increase can be observed.[16,17] The nucleogenic property of rain, particularly of the shower type, is most likely attributable to the production of ions by the splashing of droplets, the socalled Lenard-effect.[15]

Probably the most obvious physicochemical effect of the aerosol is the corrosion of exposed surfaces caused by the chlorine, sulphur, and ammonia compounds of which the nuclei consist. Manifestations of this corrosion can be observed on house facades[4] in all industrial regions with their abundance of sulphuric-acid suspensions, or at the seashore, where the relatively high salt content of the air often proves destructive of inadequately protected mechanical devices, such as camera shutters and watches. Notwithstanding the undesirable chemical properties of nuclei, they have also beneficial aspects. The bracing effect of the salty ocean air is well known to the vacationist. Also the sulphur content of the air is becoming an increasingly important natural source for this element, an essential item in soil fertility.[7] Dust, likewise, has its dual aspects which become immediately apparent when one considers the "dustbowl" of the U.S. and the fertile loess deposits in Eastern Asia. The dust from combustion processes and traffic found in city air is, for the most part, a public nuisance. However, most of the substances of which this dust is composed, such as carbon, flyashes and minerals, are chemically too inert, and the large particles that might cause erosion of surfaces, are too few to equal the importance of nuclei in the problem of air pollution. The tarry components of dust have probably a deleterious effect on plant life. However, even this growth-impairing effect is small in comparison with that of other environmental factors such as lack of fertile soil, inadequate supply of precipitation due to pavements and snow removal, and lack of sunlight because of the high buildings. This contention is borne out by the fact that in spite of existing air pollution, vegetation flourishes in city parks or wherever ample space is provided. However, space is at a premium in industrial cities so that vegetation is generally scarce. This is unfortunate because trees and shrubs are effective filters of the aerosol.

In closing, a few remarks may be made on the possible biological effects of the aerosol on human beings. Here, the reduction of sunlight, particularly its ultraviolet component, by both dust and nuclei comes foremost to the mind. This reduction, however, cannot be considered as a serious disadvantage from the physiological point of view for the following obvious reasons: 1) most city residents spend most of their daytime hours either indoors, or in subways, busses, streetcars, automobiles; 2) even in the absence of air pollution, sunlight in city streets is greatly reduced by the shade caused by city buildings; 3) even in completely unimpaired sunshine, an effect of the actinic rays could not reasonable be expected, as civilized people generally expose only an insignificant portion of their skin to the sun's rays; 4) probably the most important argument is that the beneficial antirachitic effect of sunlight has been recognized as outweighed by its detrimental carcinogenic effect, which causes a high incidence of skin cancer among outdoor workers.

With regard to the effect of the aerosol on the human respiratory system, very little factual data are available, although speculative inferences abound. Until the direct physiological reaction of the human body to all components of the aerosol is known, conclusions drawn from purely formal statistical correlations must be looked upon with utter suspicion. For instance, correlations between incidence of, or mortality from, respiratory diseases and mere "indices" of air pollution seem to be highly conjectural, as was emphasized by V. Conrad.[4]

One of the few problems explored is the retention of condensation nuclei in the respiratory tracts. Wigand,* Wait,* Amelung and Landsberg[2] have made a few measurements of the number of nuclei in the air inhaled and that of the breath exhaled. Their results, arranged and summarized in Table 5, show that the number of nuclei retained in our respiratory system increases with increased nuclei content of the air we breathe, even though the percentage retained decreases.

TABLE 5—*Retention of Condensation Nuclei in the Human Respiratory System for Various Concentrations of Nuclei in the Air*

Number of Nuclei per mm³ Inhaled	Average Number of Nuclei per mm³ Retained	Average Pct Retained
<50	16	66
50–100	24	37
>100	75	20

According to Landsberg[15] in city air containing 100,000 nuclei per cm³, a person inhales every minute almost a billion nuclei which, according to Table 5, means that he retains in his system some 150,000 billion nuclei in the course of a year. Even if the body did not expel any of this inhaled material, the body's weight increases due to nuclei would amount to only 15 milligrams per year. Whether or not the nuclear substances are harmful is not known, but the acrid sensation felt at times in the throat, from the sulphur acid in city air is nevertheless quite unpleasant. Some authors have also speculated that ions or electrically charged nuclei may be responsible for restlessness and psychological depression.

As regards dust, this component of the aerosol has, according to Gemünd,* less biological significance than the nuclei because of the relatively negligible number of dust particles in polluted air. Also, the dust elements, due to their larger size, will be more easily filtered out of the air by the outer parts of the respiratory tracts. These and similar opinions, though plausible, should, however, be checked by direct experiment.

In general, this field offers as yet untouched opportunities for the physiologist, physician, bacteriologist, epidemiologist, physicist, chemist, meteorologist, and engineer to combine their knowledge and skill in fundamental research on the biological effects of atmospheric pollution.

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* Quoted from Landsberg.[15]

* Quoted from Landsberg.[15]

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